TATA ENERGY RESEARCH INSTITUTE
BANGALORE

COMPREHENSIVE ENERGY AUDIT REPORT

AT

HINDUSTAN ZINC LIMITED
VISAKHAPATNAM

TATA ENERGY RESEARCH INSTITUTE
Krupa, 50/7, Palace Road, Bangalore 560 052

H.O

Darban Seth Block, India Habitat Centre,
Lodi Road, New Delhi 110 003

JULY 1997

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BANGALORE

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LIST OF APPENDICES

Appx No.	Description
6.0	STEAM DISTRIBUTION
6/1	Survey of Uninsulated Pipes, Flanges & Valves
6/2	Sources of Steam Leakages
7.0	CO-GENERATION
7/1	Estimation of Power Recovery Potential
8.0	COMPRESSED AIR SYSTEM
8/1	Specifications of Compressors
8/2	Compressors Energy Consumption & Running Hours
8/3	Observations on Centac Compressor
8/4	Reduction of Compressed Air Consumption in Roaster Plant
8/5	Observations of Process Air Compressor
8/6	Free Air Delivery Capacity Test
9.0	WATER PUMPING - COOLING TOWERS
9.1/1	Cooling Tower Specifications
9.1/2	Cooling Tower Performance Details
9.1/3	Roaster Plant Cooling Water Analysis
9.2/1	Major Users of Cooling Water in the Sintering Section
9.2/2	Performance Evaluation of Cooling Towers - Lead Plant
9.2/3	Once Through System of Cooling Water Circuit - Lead Smelter Plant
9.3/1	Cooling Tower Details D G Power House
10.0	PUMPS, FANS AND BLOWERS
10/1	Loading Pattern of Pumps, Fans and Blowers
11.0	ROASTER PLANT
11/1	Specifications of Roaster
11/2	Monthwise Details of Zinc Conc. Roasted, LDO Consumption, Running hours of Roaster, Preheating Burners and Lancers
11/3	Quantification of Surface Heat Losses of Roaster
11/4	Design and Observed Temperatures At Various Process Equipments
11/5	Design and Observed Pressures At Various Process Equipments
11/6	Surface Heat Losses from Waste Heat Boiler
12.0	SULPHURIC AIC PLANT
12/1	200 TPD Sulphuric Acid Plant Equipment Features
12/2	Monthwise Production Details of 200 TPD & 50 TPD Sulphuric Acid Plant
12/3	Monthwise Energy Consumption and Running Hours
12/4	Heat Recovery from 200 TPD H_2SO_4 Plant Preheater
12/5	Substitution of LDO by Furnace Oil in Preheater of 200 TPD H_2SO_4

Appx No.	Description
13.0	LEACHING AND PURIFICATION PLANT
13/1	Estimation of Steam Consumption for Neutral Pachukas
13/2	Estimation of Indirect Steam Usage in Leaching and Purification
13/3	Estimation of Surface Heat Losses from Neutral Pachukas
13/4	Possible Energy Savings By Reducing Evaporation Losses from Dorr Thickner
14.0	ZINC ELECTROLYSIS PLANT
14.1	CELL HOUSE
14.1/1	Monthly Production Vis-a-Vis Power Consumption
14.1/2	Measurement of Individual Cell Voltages - Circuit X-22
14.1/3	Measurement of Individual Cell Voltages - Circuit X-12
14.1/4	Measured Millivolt Drops Across Junctions of Busbars from Rectifier Output to Cell House Busbars
14.1/5	Measured Millivolt Drops Across Anodic and Cathodic Joints
14.1/6	Quantification of Power Loss Due to Millivolt Drop Across Anodic and Cathodic Joints
14.1/7	Measurement of Millivolt Drop Across Busbar Joints on Cell Top and Bottom
14.1/8	Power Loss due to Resistances of Anode and Cathode Electrodes
14.1/9	Power Loss Due to Electrolyte Resistance in the Cascade
14.1/10	Observations on Feed and Spent Electrolyte of Selected Cells in Cell House
14.1/11	Observations on Spent Electrolyte Coolers
14.1/12	Electrolysis Plant - Break-up of Power
14.2	ZINC MELTING FURNACE
14.2/1	Observations on Zinc Melting Furnaces
14.2/2	Theoretical Power Requirement for Zinc Melting
14.2/3	Surface Heat Losses from Russian and Ajax Furnaces
15.0	LEAD PLANT
15/1	Lead Smelting - Process Flow Chart
15.1	SINTER MACHINE
15.1/1	Monthwise Energy Consumption and Production in Sinter Machine for the year 1994 - 95
15.1/2	Input Materials to the Furnace
15.1/3	Observed parameters in Sintering Furnace
15.1/4	Heat Balance of Sinter Machine
15.1/5	Utilisation of Heat in Recirculation Air for Preheating Combustion Air
15.1/6	Use of Furnace Oil in Sinter Machine
15.1/7	Combined Efficiency Evaluation of Fresh Air and Recirculation Blower in Sintering Furnace

Appx No.	Description
18.0	DIESEL GENERATORS
18/1	Specifications of Diesel Power House Engine
18/2	Energy Consumption & Power Generation Details
18/3	Details of Diesel Generator Running Hours
18/4	Electrical Loading Parameters of 5 MW DG Sets
18/5	Observations on Performance of Diesel Engines
18/6	Operating One 5 MW DG Set Continuously - Synchronised With EB Supply
18/7	Potential Waste Heat Recovery From Diesel Generators
19.0	LIGHTING SYSTEM
19/1	Distribution of Light Fittings
19/2	Lux Level Measurements
19/3	Observations of Lighting - Kept Switched on Even During Day Time (In Outdoor Yard/Areas)
19/4	Replacement of Incandescent Lamps by HPSV Lamps
19/5	Replacement of HPMV 250 W by 150 W HPSV
19/6	Replacement of HPMV by HID Metal Halide Lamps
19/7	Use of Voltage Controllers in Different Areas for Lignting
20.0	CADMIUM PLANT
20/1	Monthwise Cathode Production and Power Consumption
20/2	Measured Cell Voltages and Millivolt Drops
22/1	List of Suppliers and Retrofits

I

EXECUTIVE SUMMARY

EXECUTIVE SUMMARY

1.0 INTRODUCTION AND ENERGY SYSTEMS

This section presents a broad summary and perspective of the important features arising out of Energy Audit study. Details are contained in the Main body of the report.

2.0 BRIEF ANALYSIS OF STUDY AND FINDINGS

Electricity, LDO, F.O, Hard Coke, LPG & HSD are the sources of energy. The plant has purchased about Rs.2,673.1 lakhs worth electricity and Rs.5,184 lakhs worth thermal energy in Lead and Zinc plant during the year 1994-95. The cost of energy in the operating expenses was around 20.0 % for the year 1994-95.

A detailed study, measurement and analysis was carried out by a team of engineers in the following areas:

* Electricity receiving, distribution and generation including utilisation areas like drives and lighting
* Steam and Air compressor generation, distribution and utilisation
* Zinc electrolysis plant
* Roaster, Leaching and acid plants
* Lead, Zinc oxide plants
* Other utilities and Process plants

The energy audit study conducted at M/s. HZL, Vizag, Smelter has identified several measures for energy optimisation by efficient utilisation and fuel switching options. The study has highlighted an annual energy saving potential of 395 kL of LDO fuel and 8.1 lakh kWh of electricity. The annual reduction on cost of above energy systems account for Rs.71 lakhs with a capital investment of Rs.73 lakhs (approximately). The areawise study and potential energy savings are highlighted in sections detailed below :

3.0 ELECTRICAL SYSTEMS

A. Transformer Load Management

i. 2 x 10/12.5 MVA, 33/6.6 kV Transformers

By operation of one transformer on load for other plant loads (apart from rectifiers) may be continued (as is practiced presently). The above proposal may be implemented during non-monsoon months (6 months in a year).

This measure is expected to yield an annual energy savings of 17385 kWh (ie., Rs.66,063) with immediate payback period. For details refer Chapter 3.6 B for details.

ii. 6.6 kV/433 V Distribution Transformers

One distribution transformer each (1600 kVA rating) at Lead plant, Utility S/s and zinc oxide plant is proposed to be switched off. The loading of individual plants should be continued with the available transformation capacity.

This measure is expected to yield annual energy savings of 36288 kWh (i.e, Rs.1,37,895), with marginal expenditure for releasing the transformers. For details refer Chapter 3.7 (A) for details.

4.0 ELECTRIC MOTORS

A. Replacement of Standard Induction Motors by High Efficiency Motors

Standard induction motors should be replaced by high efficiency motors. This measure can yield an annual energy savings of 70740 kWh (i.e, Rs 1,79,675/-) with a simple payback period of 4 years. Refer Chapter 4.2 (C) for details.

B. Optimum Sizing of Grossly Underloaded Motors

The acidic and non-acidic fan motors of Roaster plants can be optimally sized the motors are loaded only upto 13.5% and 15.9% of their rated capacity respectively. Implementation of this measure can save about 8640 kWh per annum worth about Rs.21,940/- with immediate payback. Refer Chapter 4.2 (F) for details.

5.0 STEAM GENERATION

A. Replacement of LDO by FO

In the present working of boiler, LDO is being used as fuel. It should be substituted by Furnace Oil. Implementation of this measure is expected to result in annual savings of Rs.26.61 lakhs with a simple payback period of 0.94 year. Refer Chapter 5.3.1 B for details. TERI has suggested few more parties for taking up replacement of burners.

6.0 STEAM DISTRIBUTION

A. Insulation of Steam Lines

Insulation of uninsulated steam mains, flanges and valves should be taken up. This has the potential annual energy savings to the tune of 1.565 kL LDO (i.e, Rs.11,440/-) with a simple payback of 1.6 years. Refer chapter 6.2 A for details.

B. Steam Leakage

Existing steam leakages as identified should be plugged at the earliest An annual energy savings to the tune of 7.36 kL of LDO (i.e, Rs.63,290/-) yields simple payback period of 0.4 years. Refer Chapter 6.2 B for details.

7.0 WATER PUMPING AND COOLING TOWERS

A. Once Through Cooling Water Pumping System in Smelter Plant

It is preferable to have once through pumping system to eliminate two hot well pumps. Annual energy savings to the tune of 1.408 lakh kWh (i.e., Rs.5 35 lakh) can be envisaged with an investment of Rs.8.00 lakhs giving a payback of 2.5 years. Refer Chapter 9.2.2 C for details.

8.0 ZINC ELECTROLYSIS PLANT (CELL HOUSE)

8.1 CELL HOUSE

A. Specific Energy Consumption and Monitoring of Cell Voltages

i. Presently the specific energy consumption of electrolysis plant is made based on available AC metering systems. However precision metering systems on AC and DC systems give the actual values of hourly energy consumption for meaningful analysis. Microprocessor based metering should be installed

ii At present there are no cell voltage monitoring systems A common instrumentation panel indicating all the cell voltages and bus-bus voltages may be installed for better supervision and control

B. Anodic and Cathodic MilliVolt Drops

The copper bus to anode and cathode voltage drops are observed to be on the higher side This varies from 15 mV (min) to 187 mV (max)

From the observed values, a tolerance millivolt drop of 30-40 mV can be a guiding factor for further control of voltage drops

By regular monitoring, gap adjustment and supervision, it is possible to minimise these voltage drops by 10% and hence minimise power losses Reference to Sec 14 2 3 gives trials conducted on cascade nos 5,17,20 and 27

By improved supervision and monitoring by operators, it is possible to minimise losses and achieve savings on a continued basis

Power loss in X-12 circuit/h = 165 kW
Power loss in X-22 circuit/h = 183 kW

Implementation of this measure yields a annual energy savings to the tune of 2,50,560 kWh (i.e Rs.6,36,850) with a simple payback period of 4 months. Refer Chapter 14.2.4 (d) for details.

C. Cascade Bus to Bus Series Milli Volt Measurements

The mV drop values of bus to bus joints should be brought to average values by regular cleaning, and measurement (monitoring). It is estimated that 30% reduction in total mV drop can be reduced in X-22 circuit and 50% mV drop may be reduced in X-12 circuit. This measure can yield an annual energy savings to the tune of 1,29,074 kWh (i.e, Rs.3,27,840) with a simple payback period of 3 months. Refer Chapter 14.2.4 (b) for details

9.0 LEAD PLANT

A. SINTER MACHINE

i. Use of Furnace Oil in Place of LDO in Sinter Machine

There exists a good potential in cost savings by using furnace oil instead of LDO in burner. By implementing this measure, the cost of hourly oil consumption can be reduced to Rs.307/- from existing Rs.430/-. This measure has an annual energy saving potential of Rs.8.856 lakh kWh with out any investment. Refer Chapter 15.1.2 G.

ii. Metering of Oil Consumption

Metering of oil consumption should be practised and monitored regularly. Consumption metering can be done by using either dipstick method or by using flow meter

B. REFINERY

i. Avoiding Heat Loss Through Door Openings

To avoid heat loss through door opening, measures should be initiated towards provision of peep holes for checking flame and closing the door during furnace operation. Implementation of this measure is expected to yield annual energy saving to the tune of 50.21 kL of Furnace oil (i.e., Rs.2.68 lakh) with marginal investment. (Refer Chapter 15.4.3.E for details).

ii. **Controlling Excess Air Levels in Burners**

Excess air in burner of kettle furnaces can be brought down to 30% by commissioning the ratio controllers installed already in the system. The implementation of this measure will improve the efficiency substantially. Refer Section 15.4.2 (B) for details.

10.0 ZINC OXIDE PLANT

A. Use of Furnace Oil in Place of LDO

There exists a good potential in cost savings by using furnace oil instead of LDO in burner in Waelz and Clinker kilns By implementing this measure, oil consumption can be reduced to by 256 kL/year. The annual cost savings are estimated at Rs.18.69 lakhs per annum with an investment of Rs 11.8 lakhs giving a payback of less than one year. Refer Chapter 16.3 C for details.

11.0 DIESEL GENERATOR SETS

One DG Set of 5 MW capacity should be run continuously and waste heat recovery system/vapour absorption machine should be installed for obtaining 300 TR of refrigeration load. This would suffice to meet one cooler load of spent electrolyte system. Implementation of this measure is expected to yield annual energy savings of Rs 22 72 lakhs with an investment of Rs 135 lakhs giving a simple payback period of 5 9 years Refer Chapter 18 3 C for details However this proposal becomes viable only when additional sanction of demand is put up to APSEB

12.0 LIGHTING SYSTEM

A. Replacement by More Efficient Lighting

1 The 250 W and 400 W HPMV lamps should be replaced by 150 W and 250 W HPSV lamps This measure can achieve energy savings to the tune of 57960 kWh/year amounting to Rs 2,20,248 with a simple payback period of 1.53 years. Refer Chapter 19 2 B 2 for details.

2. HPMV lamps in the central workshop should be replaced with HID. Implementation of this measure will save energy to the tune of 7680 kWh/year amounting to Rs.29,184 with a simple payback period of less than one year. Refer Chapter 19.2 B.3 for details.

B. Voltage Controllers for Lighting System

Voltage controllers/Energy savers should be installed in the lighting system which will save energy to the tune of 93239 kWh/year amounting to Rs.3,54,798. The cost of implementation being Rs.4.12 lakhs giving a simple payback period of 1.18 years. Refer Chapter 19.2 C for details.

SUMMARY OF POTENTIAL SAVINGS

Sl No	Area/Section	Estimated savings in energy consumption				Total savings per year Rs	Cost of implementation Rs	Simple payback period Yrs
		Electrical energy		Thermal energy				
		kWh/year	Rs /year	LDO kL/year	Rs /year			
1	Electrical Systems	53 673	2 03 958	-	-	2 03 958	Nil	Immediate
2	Electric Motors	80 970	2 75 000	-	-	2 75 000	7 24 000	2 63
3	Steam Generation	-	-	130	9 50 000	9 50 000	25 00 000	2 6
4	Steam Distribution & Utilisation	-	-	8 925	74 730	74 730	43 000	0 60
5	Water Pumping & Cooling Towers	1 40 896	5 35 000			5 35 000	8 00 000	1 5
6	Zinc Electrolysis Plant	3 79 634	14 42 580			14 42 580	6 00 000	0 4
7	Lead Plant			1/1 36	11 52 800	11 52 800		Immediate
8	Zinc Oxide Plant			256	18 69 000	18 69 000	11 80 000	0 63
9	Lighting System	1 59 006	6 04 222			6 04 222	7 83 900	1 29
	Total	8,14,179	30,60,760	394 925	40,46,530	71,07,290	72,80,900	-

* Quantified in terms of LDO

MAIN REPORT

TATA ENERGY RESEARCH INSTITUTE
BANGALORE

HINDUSTAN ZINC LIMITED
ZINC SMELTER
VISAKHAPATNAM

COMPREHENSIVE ENERGY AUDIT REPORT

1.0 INTRODUCTION

This report presents the findings of Energy Audit of M/s. Hindustan Zinc Limited, Visakhapatnam, Andhra Pradesh.

Energy Audit study was carried out during July 95 to Aug 95 in the following areas to identify energy saving opportunities.

- Electrical System
- Electrical Motors
- Steam Generation, Distribution and Utilisation
- Compressed Air Generation, Distribution and Utilisation
- Water Pumping System
- Cooling Towers
- Pumps, Fans & Blowers
- Roaster Plant
- Sulphuric Acid Plant
- Leaching & Purification Plant
- Zinc Electrolysis & Melting Plant
- Lead Plant
- Zinc Oxide Plant
- Chilling Compressor
- Diesel Generator &
- Lighting System

During the study every attempt was made to understand the operational features and working of the project in the proper perspectives. For purposes of analysis, the various operations were observed, relevant data collected, measurements taken wherever necessary using portable instruments. There was constant interaction with the plant personnel who gave full support to the Study Team.

This report presents the analysis, findings and recommendations for achieving energy savings.

2.0 ENERGY CONSUMPTION PROFILE

2.1 PRODUCTION PROFILE

The Plant produces metals like Zinc ingots, Lead ingots, Sulphuric acid, Cadmium and Silver.

Details of installed capacities of various products & byproducts and actual production for the past three years are shown in Appendix - 2/1.

Capacity utilisations for the past three years in respect of various products and byproducts have been tabulated below :

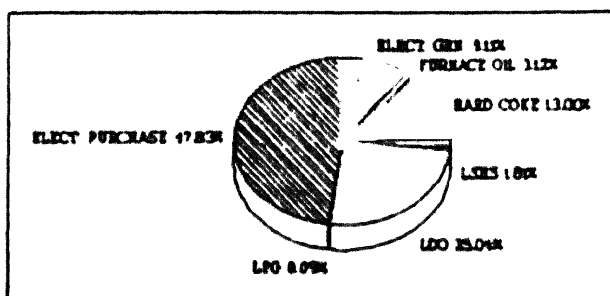
Year	% Capacity Utilisation				
	Zinc	Lead	H_2SO_4	Cadmium	Silver
1992 - 93	99.0	75.1	-	98.8	16.4
1993 - 94	100.1	11.0	76.7	61.6	37.3
1994 - 95	90.1	50.0	69.4	46.4	50.4

(Source : Annual Report)

Capacity utilisation is more than 90 % in respect of zinc ingot production and in respect of other plants capacity utilisation have varied which can be attributed to various reasons.

2.2 ENERGY SOURCES

Electricity, LDO, Diesel, Furnace Oil, LPG and Hard coke are the major sources of energy to the plant. Major electricity consumption is in Zinc Electrolysis plant. LDO is mainly used in furnace and boilers of different units in the plant. Hard coke and LPG find usage mainly in Lead plant.

Contribution of different sources of energy, in terms of percentage, for the production of Zinc and Lead for 1994-95 is shown in the pie-chart.

From the chart it can be observed that major source of energy is electricity and its contribution to the total energy input is around 56%.

The plant has a total power requirement of 27 MW and has a captive generation capacity of 22 MW.

Break-up of the Electricity purchased and Electricity generated for the past three years are given in Appendix - 2/2.

The Percentage break-up of Electricity Purchased and Generated for the past three years is given below :

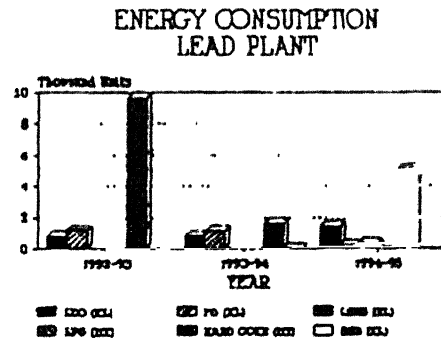
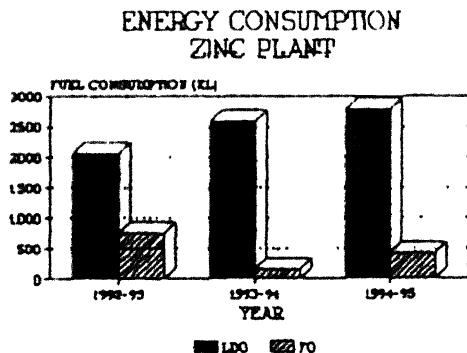
Year	% of Electricity Purchased	% of Electricity Generated
1992 - 93	69.00	31.00
1993 - 94	82.40	17.60
1994 - 95	84.00	16.00

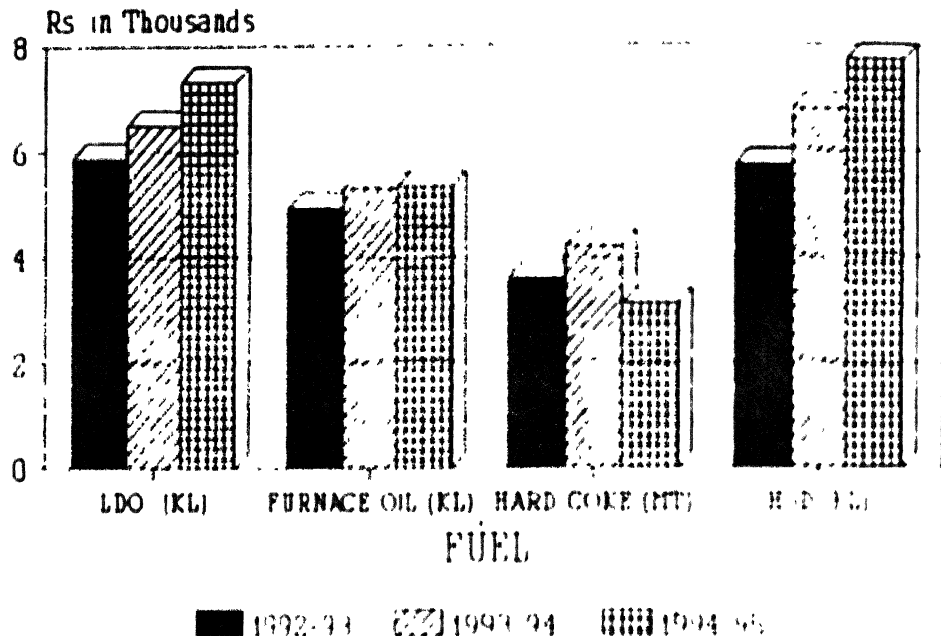
It can be observed that purchased electricity has progressively increased over the years with a corresponding decline in self generation.

Forms of energy usage in Zinc and Lead plant are depicted below and the same is tabulated in Appendix - 2/2.

ENERGY CONSUMPTION
ZINC PLANT

ENERGY CONSUMPTION
LEAD PLANT

YEAR

From the above graphs it is clear that the major energy inputs are LDO and Hard Coke, for Zinc and Lead plants respectively.

ENERGY COST
COST IN RUPEES

The production, power consumption and specific power consumption in the Zinc and Lead plant for the past three years are given below .

Year	Ingot Production (Mt)		Power Consumption (Lakh kWh)		Specific Power Consumption kWh Mt	
	Zinc	Lead	Zinc	Lead	Zinc	Lead
1992 - 93	29702.50	16532.0	1253 70	78 29	4221 0	473.0
1993 - 94	30040.00	2415.00	1265.28	17 26	4212.0	715.0
1994 - 95	27025.00	11003 0	1141.26	83.52	4223.0	759.0

The specific energy consumption figures for the year 1994 - 95 are 4223.0 and 759.0 against standard set figures of 4175.0 and 553.0 for Zinc ingot and Lead ingot respectively. The figures show actual energy consumption by the plant more than the standard set figures. One of the factors contributing to the extra consumption may be due to under utilisation of plant capacity.

The cost of energy in the operating expenses of the plant is around 11.1%, 20.4% & 20.0% for the years 1992 - 93 to 1994 - 95 respectively. The element of cost in the production of Zinc and Lead is given in Appendix - 2/3.

The trend in fuel cost over the past 3 years has been shown in chart below and the same are tabulated in the Appendix - 2/3.

From the above graph the percentage variation in fuel prices had been calculated taking the year 1992-93 as base year and the same are tabulated below :

Energy Source	1993-94	1994-95
LDO	+ 10.6	+ 24.5
F.O.	+ 9.6	+ 8.0
Hard Coke	+ 16.7	- 13.4
HSD	+ 18.2	+ 34.3
Purchased Electricity	+ 11.3	+ 21.1

It may be seen that prices of FO, LDO, HSD and electricity have significantly increased vis-a-vis Hard coke which have shown a declining trend.

3.0 ELECTRICAL SYSTEMS

3.1 SYSTEM DESCRIPTION

The plant receives power supply from APSEB at 132 kV, on two lines, via 'moose ' conductors. Two power transformers of 35-40 MVA capacity supply, 33 kV bus from where loads are tapped to rectifier and plant transformers. From 33 kV bus, the plant HT motors and distribution transformers are supplied by 2 x 10/12.5 MVA transformers at 6.6 kV bus (with capacitor banks connected).

Three DG sets of 5 MW are connected to 33 kV bus through step up transformers. Three MW DG sets (2 Nos.) are connected to 6.6 kV bus for black start. Schematic diagram of power supply receiving and distribution arrangement is shown below :

* Bus coupler at 33 kV and 6.6 kV are normally kept closed.

Plant Power Requirements

Contract Demand kVA	Max.demand kVA	Monthly consm. kWh (Range)	Avg monthly PF
22000	21840	1,25,00,000 to 1,45,00,000	Above 0.95

System Transformer Details

Sl No	Equipment	Rating	No of systems	Make
1	I/C power transformer	35/40 MVA 132/33 kV	2	NGEF
2	Rectifier transformer	9460 kVA 13 kA=620V	2	NGEF
3	Power transformer	10/12 5 MVA 33/6 6 kV	2	Bharat Bijlee
4	Distribution transformer	6 6 kV /433 V 1600 kVA 1250 kVA 1000 kVA	 12 3 3	GEC
5	a DG sets Stepup transformer b DG Sets	5 MW, 11 kV 7 5/10 MVA, 11/33 kV 3 5 MW, 6 6 kV	3 3 2	Allen-UK Bharat Bijlee Russian

3.1.1 132/33 kV SUBSTATION

The plant 33 kV bus is supplied by two numbers of 35/40 MVA 132 kV/33 kV incoming power transformers and rectifier loads (15 MW) form the major load on 33 kV bus.

Earlier the plant was receiving power at 33 kV from APSEB, and during 1990, these two power transformers were installed on the insistence of APSEB to receive power at higher voltage. The transformers are having OLTC with 17 taps and range of ±10% on primary.

The incoming breakers are of SF6 type 1250 A capacity with necessary isolators. Metering of system energy is by "Duke Arnics" make trivector meter installed by APSEB.

The 33 kV secondary side in the yard has two isolators and a bus coupler. The outdoor yard houses the 10/12.5 MVA 33/6.6 kV power transformers (2 No.) and 7.5/10 MVA, 11/33 kV (3 No.) generator transformers. Name plate details of power transformers are given in Appendix - 3/2.

3.1.2 M R S - 33 kV SYSTEM

The 33 kV bus in MRS has four outgoing 1250 A, 1500 MVA, MOCB panels supplying power to two rectifier transformers and two numbers of 10/12.5 MVA power transformers. Three incomers from 3x5 MW DG set, step-up transformers are connected to 33 kV bus through synchronising controls. Diesel power house is situated adjacent to MRS. The name plate details of power generator transformers are given in Appendix - 3/3.

3.1.3 M R S - 6.6 kV SYSTEM

The 6.6 kV bus chiefly supplies power to all the HT motors and 18 distribution transformers of 1600, 1200 & 1000 kVA ratings. The 6.6 kV MOCB panel has three sections with two bus couplers and 630 A, 250 MVA MOCB's for power delivery to HT motors, transformers and 4 banks of 2016 kVAr capacitors. The capacitor banks are situated below the basement of MRS building and 6.6 kV/433 V transformers are situated at load centres.

DG sets of 3.5 MW, 6.6 kV (2 Nos.) are directly connected to 6.6 kV bus for black start operation.

The name plate details of distribution transformers, capacitor banks are given in Appendix - 3/4 and 3/5 respectively.

3.2 POWER TARIFF

Power billing is by two part tariff; Metering of energy parameters is at 132 kV ODY and APSEB has used "Duke Arnics" trivector meter. Plant has Landys and Gyr meter installed for reference on 132 kV panel.

Demand charges/ kVA	Energy Cost kWh	Energy Cost/kWh*
Rs.110	Rs.2.35	Rs.2.54

* includes fuel surcharge

3.3 EXPANSION PLANS

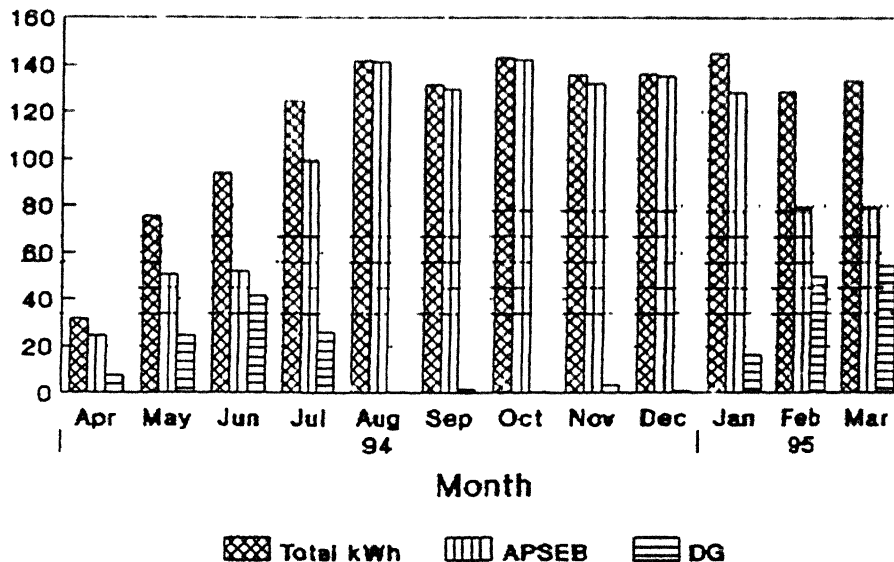
The plant is proposing for expansion in capacity of electrolysis plant. Additional cascade is planned for installation, thereby requiring increase in installed capacity of rectifier transformers to 13.5 MVA. Plant has approached APSEB for additional 3 MVA contract demand. Additional auxiliary load growth is expected to be at 250 kW (approx.).

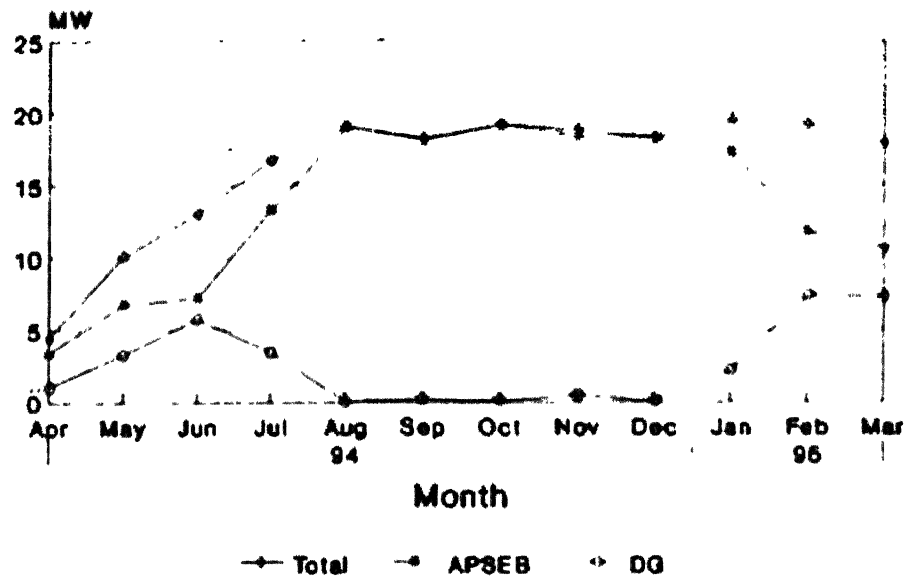
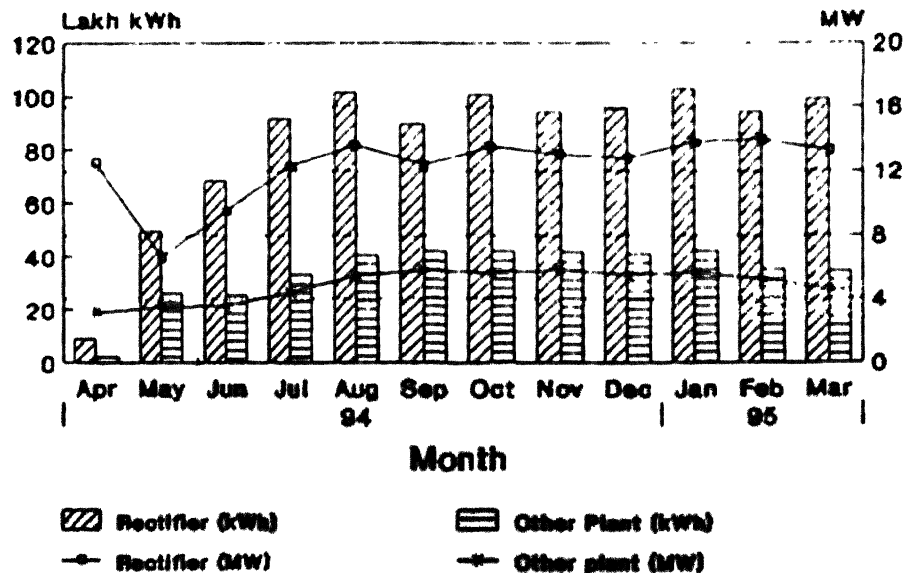
However, the installed capacity of DG set ie., with a maximum generation potential of 13 MW (under existing DG conditions) is utilised presently against power cuts (upto 40% demand cut) and during load shedding/power shutdowns.

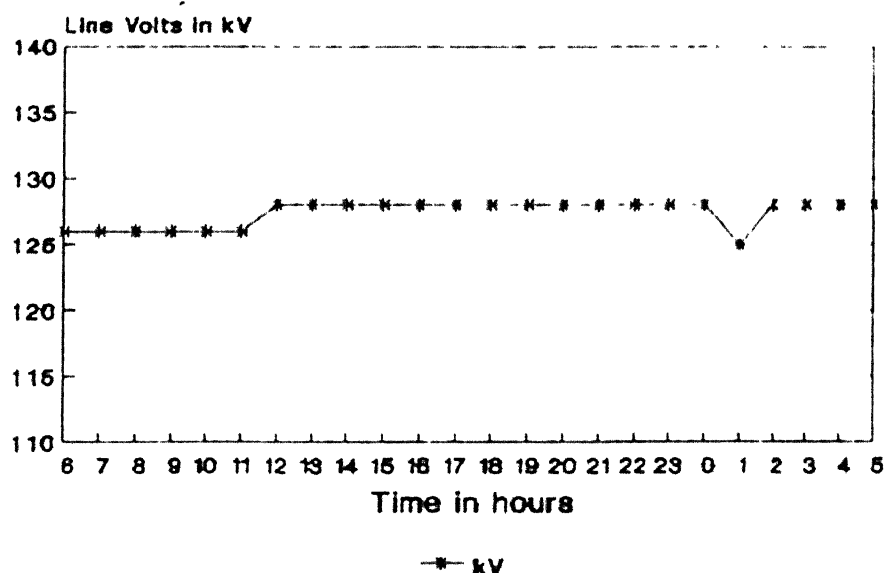
3.4 GENERAL OBSERVATIONS

For the purposes of study, measurements were taken using various meters and panel instruments. Measurements were also taken using kW/Cosϕ digital instruments by conversion of CT/PT test terminal jacks.

Readings from APSEB trivector meter and daily logbooks, past records are taken and analysed. Analysis of loading conditions and computations are done using necessary PCAT/mini systems.

Electricity consumption and average load requirements of the plant are shown in Appendix - 3/1.

It is observed that the total monthly electricity requirement varies from 124.7 (min.) to 144.76 lakh kWh (max.), (when plant production is stable).

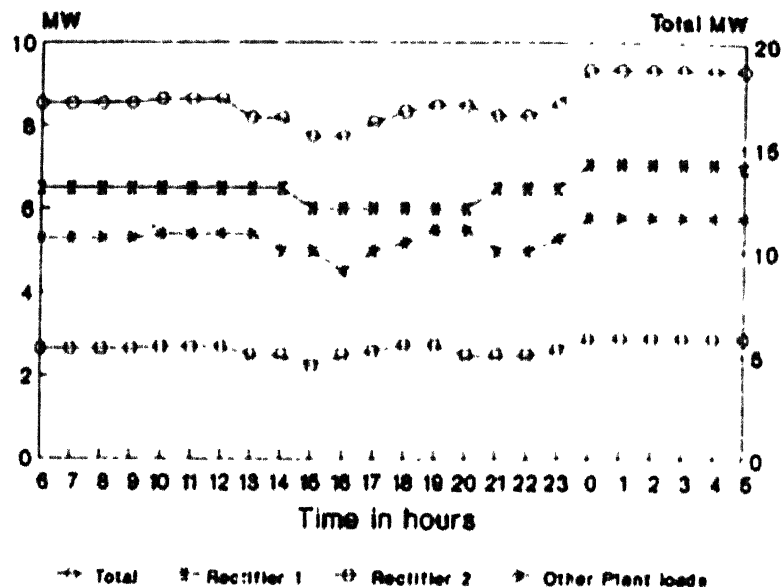
The power requirements of the plant is 17-19 MW. The variation in average power requirements and contributions from APSEB and DG are also shown in the line graph below

The average consumption of rectifiers and other plant loads are depicted in the following graphs.

3.5 SYSTEM LOAD PARAMETERS

The load parameters of the plant are placed in Appendix - 3/7 for a typical day (10-08-95). The incoming voltage levels were varying from 125 kV to 128 kV. The OLTC was observed to be operating at position 16 or 17 to give secondary voltage of 33 to 33.5 kV recorded from panel meters. The OLTC's of the two power transformers (operated in parallel at present) are having master/follower combination.

for a typical day

The incoming system PF, current and frequency levels are given below for the typical day referred.

Power Factor	132 kV Current (A)	Frequency (Hz)
0.98-0.99	40	47.9 - 48.2

The monthly record of power failure and power restrictions imposed by APSEB (peak load restrictions) is given in Appendix - 3/6. It is observed that power failure period during 1994-95 is minimal, however, DG sets are run whenever APSEB load restrictions are imposed.

The MW load requirement of the plant over 24 hours is depicted in the curve below with a break-up of requirement by rectifiers 1, 2 and other plant loads. (Refer Appendix - 3/7 for details).

For a typical day

Details of calculations of system loading are shown in Appendix - 3/8.

	System factors	Data
132/33 kV system	Annual load factor (132/33 kV system)	65.6%
	Annual loss load factor	0.440
33/6.6 & 6.6 kV/ 433 V system	Annual load factor	86.99%
	Annual loss load factor	0.756

3.6 TRANSFORMER LOAD MANAGEMENT

A. 35/40 MVA Power Transformer

The two incoming transformers are operated in parallel and average plant load is around 19.11 MVA when one transformer is taken out of service for maintenance full load is taken on one transformer only.

Additional demand requirement is projected to an extent of 3000 kVA due to expansions of electrolysis plant. With the above loads in service, the loading of the transformers will be upto 31.5% which is close to optimum loading level of 33.2%.

Calculation of losses with 35/40 MVA, 132/33 kV transformers

Annual energy losses	2x35/40 MVA on load (present practice)	1x35/40 MVA on load with proposed expansion in load	Operating 1x35/40 MVA under existing loading conditions
% load	27 3	63 0	54 6
N/L loss kWh	315360	157680	157680
Load loss (kWh)	124962	251287	187721
Total transformation losses (kWh)	440322	408967	345401
Savings in transformation losses (kWh)	-	31354	94920

The savings due to the above proposal under the present load conditions will be 31354 kWh/annum. Details are worked out in Appendix - 3/9. However, considering the marginal savings and reliability of power supply requirement, it is proposed to continue loading both the transformers.

B. 10/12.5 MVA Power Transformer

Presently only one power transformer of 33/6.6 kV, 10/12.5 MVA power transformer is on load. This has resulted in saving transformation losses to an extent of 25853 kWh per annum. Details are worked out in Appendix - 3/10.

Loading of 33/6.6 kV transformers

Considering 2x10/12.5 MVA as one system

Data	2 transformers on load	1 transformer on load
Avg load in MW (%)	5.2 (26%)	5.2 (52%)
N/L loss/annum (kWh)	168192	84096
Load loss/annum (kWh)	49317	98634
Total losses/annum	217509	182730

Savings in transformation losses/annum = 34770 kWh

It is recommended to load one transformer during non-monsoon months (6 months) in a cyclic rotation of one week.

Annual savings in energy losses = 17385 kWh
(for 6 months in a year)

3.7 6.6 kV/433 V DISTRIBUTION TRANSFORMERS

A. Present status

Plant has 18 Nos. of distribution transformers under installation for plant LT loads, on 6.6 kV bus. Distribution transformers of the plant are 20 years old and these are situated in load centres with one standby on no load. Details are given below :

Area	On load	No load
Lead plant	2	1
Zinc Oxide	1	1
Roaster & acid	2	2
Cadmium/workshop	1 + 1	-
Leaching/compressor	2	2
Electrolysis	1	1
Russian furnace	1	-

The transformer feeder details, rating & percentage loading and details of loading (voltage and current) of each transformer on load are given in Appendix - 3/10.

The summary of load on transformers are given below :

Sl No.	Ref.	Feeder details	Rating (kVA)	% loading	Estimated kVA load
1	X-30	Lead plant	1600	18-20	180
2	X-31	Zinc oxide	1600	5-10	160
3	X-41	Zinc oxide	1600	30	480
4	X-33	Lead smelter	1600	40	640
5	X-43	Lead smelter	1600	7-10	160
6	X-35	Roaster plant	1600	42.50	800
7	X-45	Roaster plant	1600	- *	-
8	X-36	Acid/cooling tower	1600	18-24	384
9	X-46	Acid/cooling tower	1600	- *	-
10	X-34	Cadmium plant	1000	10-20	200
11	X-44	Workshop	1000	26-29	290
12	X-32	Leaching plant	1600	28-35	560
13	X-42	Leaching plant	1600	17-35	560
14	X-37	Electrolysis	1600	11-18	288
15	X-47	Electrolysis	1250	44-46	575
16	X-48	Russian tr.	1000	53	530
17	X-39	Compressor	1250	5	62
18	X-49	Compressor	1250	28-32	400

* Transformers are kept idle charged.

Max.load on plant transformers = 6269 kVA

Max load at 0.8 utilisation factor (This load is apart from HT motor loads on 6.6 kV bus) = 5015 kVA

B. Analysis

Transformer Load Management with Distribution Transformers

The load centre transformers for plant LT motor and lighting loads are supplied by separate 6.6 kV power cables from MRS. The maximum total load on plant transformers was recorded to be 6269 kVA (taken on typical working days). The total load at 80% utilisation factor is 5015 kVA (This load on 6.6 kV bus is apart from 1503 kVA load of HT motors).

The total transformation capacity available for the plant is 25950 kVA of distribution of power at 433 Volts. This is on the higher side resulting in high magnetisation load contributing towards core losses.

It is also observed that some of the transformers have developed leakages.

The following data is taken up for analysis :

Sl No.	No.of transformers in area indicated	Load kW	Capacity of transformation in kVA available
1	Lead plant	980	3 x 1600 = 4800
2	Zinc oxide	640	2 x 1600 = 3200
3	Roaster/acid & cooling tower	1184	4 x 1600 = 6400
4	Leaching/compressor	1582	4 x 1600 = 6400
	Add 10%	438	-
	Total load	4824	20800

About 23.2% of total transformation capacity is utilised.

No Load loss of each transformer = 2.8 kW
Load loss of each transformer = 19.0 kW

Sl No	Data	Unit	Lead plant 3 trfrs	Zinc Oxide 2 trfrs	Roaster Acid & CT	Leaching Compressor
1	No load loss	kW	8.4	5.6	8.4	8.4
2	Load loss	kW	57.0	38.0	57.0	57.0
3	Annual No load loss	kWh	73,584.0	49,056.0	98,112.0	98,112.0
4	Annual load loss	kWh	15,715.0	10,066.0	12,919.0	23,065.0
	Total losses	kWh	89,319.0	59,122.0	1,11,031.0	1,21,177.0

Proposal:

The transformers at the following location may be switched off ON primary and secondary during non-monsoon period to save no load losses for 6 months in a year.

Sl. No.	Location	No. of Transformers to be loaded	No. of transformers proposed to be switched off
1	Lead Plant	2 x 1600	1
2	Zinc Oxide Plant	1 x 1600	1
3	Roaster/Acid & CT	3 x 1600	1

Sl. No.	Plant	System Transformation capacity available kVA	Annual No Load losses kWh	Annual Load losses kWh **
1	Lead Plant	3200	49056	23602
2	Zinc Oxide Plant	1600	24528	20133
3	Roaster & Acid	4800	73584	22968
			147,168	66,703

Savings in transformation losses/p.a = 54,432 kWh

Cost of savings per annum = Rs.1,32,563

Cost of implementation = Nil

3.8 RECTIFIER TRANSFORMERS

A. System Details

The two auto-rectifier transformers are rated at 9460 kVA each. Each unit supplies a rectifier cubicle at 514.5 Volts to 375.9 Volts. The DC output from rectifiers are taken at 13 kA and 580 to 630 Volts to electrolytic cell. The circuits are independently supplying DC power to two separate group cascades which are in series and unearthed. The cascades/cells are supported on porcelain insulator and the DC busbars are having hylam/glass epoxy insulators. The transformer details are given in Appendix - 3/12.

Both the circuits X-12 and X-22 were silicon diode type of rectifiers. The X-12 circuit has been recently changed to thyristorised version with a capacity to be used upto 15 kA. Expansion plans are underway in Electrolysis plant to replace auto-rectifier transformers with 13500 kA capacity for X-12 circuit. Similar modifications are proposed for X-22 circuit in phases.

B. Loading of Transformers on AC Side

The loading on AC side has been observed to be varying from 5.0 MW to 7.0 MW at 0.92 to 0.98 PF lag.

	AC (kV)	AC (Amp)	MW	PF	MVA*
X-11	33	115	5.8	0.92	6.30
X-12	33	130	7.0	0.98	7.14

* Calculated

The loading depends on the number of cascades in service and the number of cathodes/anodes working in each cell.

The transformation losses in the AC system works out to 4.5 to 5.0 lakh kWh/annum in each circuit This forms about 1% of total Electrolysis plant power consumption. There is a proposal to replace the diode rectifier system to thyristorised one for the second circuit and this is an energy saving measure. Details are given in Chapter 14.2.2.

B. DC Distribution

The DC distribution from each rectifier is at 12 kA - 13 kA and 580 to 620 Volts. Measurements of DC distribution in Electrolysis plant are dealt in detail in Section 14.0. The measurement of rectifier, cascade and cell circuit voltages are given in Appendix - 14/1.1 to 14/1.12. The quantification of various losses in distribution systems are given below for one circuit.

Loss area	Power loss in kW
DC distribution of bus to bus	29.3
Intercell anode-cathode loss	165

The recommendations towards minimising DC distribution losses are dealt in Section 14.0.

3.9 SYSTEM PF MANAGEMENT

Plant has installed four capacitor banks of 2016 kVAr each on 6.6 kV bus. The average monthly PF of incoming system is observed to be maintained above 0.95 (0.96 during July 1995).

The APSEB trivector meter readings taken for 2 hours are given in Appendix - 3/13.

Plant has installed capacitor banks on 6.6 kV panel. Details of these are given in Appendix - 3/5. Measurements of plant power system parameters were made using portable kW/Cosϕ meter (6.6 kV/110 V meter). Details are given in Appendix - 3/14 and 3/15.

It is generally observed that 3 to 4 capacitor banks are switched 'on'. When the rectifier loads are shutdown/taken online, the capacitor banks are also switched 'off' and 'on', by observing the incoming 132 kV power factor.

Analysis

The rectifier loads are operating at 0.93 to 0.98 pf (instantaneous values). The other plant loads supplied through distribution transformers account for 5015 kVA and 1503 kVA of HT motor loads operating at pf of 0.55 to 0.71. The plant auxiliary loads (other than rectifiers) have recorded variations as below :

Plant loads	High	Low	Average
HT motors & loads on auxiliary tr.	6.2 MW	4.2 MW	5.1 MW

During study and measurements, the following observations were made :

Total load on plant = 19.6 MW

PF of incomer = 0.97 lag

Auxiliary plant load = 4.7 MW

PF on 6.6 kV bus = 0.96 lag

Effective kVAr O/P = 3112

Load on 2 DG sets operating at 33 kV bus = 6.4 MW

Reactive power generation = 2.73 MVA (approx.)

During power cuts imposed from APSEB, one or two DG sets are run to meet the short fall. It is observed that reactive compensation available at 33 kV and 6.6 kV bus (with two banks) are optimal. Details of measurements are given in Appendix - 3/14 and 3/15.

The 433 Volt bus PF is observed to be very low, ie., PF measured on MCCs were recorded between 0.5 to 0.8. To improve the load bus PF and minimise cable heating/ distribution losses, it is proposed to install 550 kVAr LT capacitor banks at power/motor control centres in the plant.

However since P.F. compensation is already achieved at 6.6 kV bus, this was not found to viable, considering space required at L.T. room, additional building cost and maintenance problem. However, Capacitor banks of 2x200 kVAr are installed at Acid plant LT, but are kept out of circuit. These banks may be dismantled and capacitor units may be installed at load centres. The above proposal may be tried out at laod centres having low p.f. to minimise losses in distribution and I^2R losses in transformers.

3.10 BUS VOLTAGE CO-ORDINATION

A. 132 kV & 33 kV Bus System

The incoming voltage conditions are satisfactory. During summer, it is observed from records that lowest voltage recorded is around 122 kV. However OLTC on 2x35/40 MVA, 132/33 kV transformers with master/follower circuit are set to correct the secondary bus voltage to 33.5 kV level. This is satisfactory for 33 kV bus since about 13 - 14 MW loads are supplied to auto rectifier transformers. It is recommended to operate 33 kV bus at 34 kV level with a view to minimise transformation and distribution losses in the Electrolysis plant circuit. At this recommended voltage level, the auto-transformer of rectifier transformer takes control of secondary voltage levels for rectification.

B. 6.6 kV Bus System

The 6.6 kV bus voltage is maintained at 6.3 kV by the operation of on-load tap changer of 33/6.6 kV, 10/12.5 MVA Bharat Bijilee transformer. Trials were conducted to observe the pf and voltage levels of the bus by switching 'on/off' the capacitor banks.

However, after the installation of capacitor banks, the 6.6 kV bus voltage is observed to be at 100% value when two or three banks are in service, due to raise in bus voltage. The loads chiefly comprise of HT/LT induction motors and underloaded transformers. From Appendix - 3/15, it may be observed that capacitor banks have improved the bus voltage from 6.3 kV to 6.57 kV with two banks in service.

It is recommended to operate 6.6 kV bus at 6.5 kV level by suitably setting the OLTC of 10/12.5 MVA, 33/6.6 kV power transformer. The busbar and HT cable losses of motors/transformers can be minimised by the above measure. However, the off-load settings distribution transformers must be properly adjusted and the same is dealt in next section.

C. LT - 433 Volts Bus

Plant has 18 distribution transformers with off-load tap changers operating either on-load or as standby. Observed voltage levels and distribution transformer off-load taps are observed to be fixed at positions ranging from 3 to 1. Secondary voltages are in the range of 410-450 Volts, which is on the higher side.

Details are given in Appendix - 3/16.

In view of recommendations at Sec.B, above to adjust the 6.6 kV bus to 6.5 kV (from existing 6.3 kV), it is recommended to change the tap down by one step as shown below :

Existing transformer tap	Proposed change of tap position
3	3
2	2
1	1

Appendix - 3/17 gives the observations made on ET plant load feeders for measuring the voltage drops. The load are extended through LT cables to a distance of 400 m from leaching/compressor substation.

It is analysed that necessity of providing separate transformer for a ET plant loads does not exist and the present installation is satisfactory with respect to voltage levels.

3.10 Distribution losses

The plant has vast HT/LT cabling network to supply plant rectifiers, HT motor and distribution transformers. The data of HT cables were made available and the HT cables used for the loads are adequate and the line losses are within limits. Details of the calculation are given in Appendix 3/21.

The transformer LT panels are situated in load centres with a maximum LT cable distance of 300 - 400 m. The average loads on the transformer are also within 20-50%. By and large the cable sizes provided for LT drives are adequate. Plant has used 240 Sqmm cables in multiple runs at all load centres. Details are given in Appendix 4/5.

The overall distribution losses of plant are estimated to be around 0.5% of annual consumption.

3.11 RECOMMENDATIONS

Based on studies made in Section 3.0, the following recommendations are made for minimising energy losses:

A. Transformer Load Management

i. 2 x 10/12.5 MVA, 33/6.6 kV Transformers

The operation of one transformer on load for other plant loads (apart from rectifiers) may be continued (as is practiced presently). This proposal should be implemented during non-monsoon months (6 months in a year).

Implementation of the above measure has resulted in minimisation of losses. Details are given in Appendix - 3/10.

Annual savings in transformation energy losses = 17385 kWh

Annual cost of energy savings = Rs.66063/-

Cost of investment = Nil

Payback period = Immediate

ii. 6.6 kV/433 V Distribution Transformers

One distribution transformer each at Lead plant, Zno plant and utility S/s is proposed to be switched off from circuit. The loading of individual plants can be continued with the available transformation capacity.

Techno-economic details are given in Appendix - 3/11.

Implementation of the above measure for 6 months in a year is expected to yield energy savings as given below :

Annual saving in transformation losses	=	36288 kWh
Cost of annual savings	=	Rs.137895/-
Cost of implementation	=	Nil
Payback period	=	Immediate

3.12 SUMMARY OF POTENTIAL SAVINGS

Sl No	Recommendations	Energy savings kWh/yr	Cost savings Rs./yr	Cost of implementation Rs lakhs	Simple payback period (yrs)
1	**Transformer Load Management** i) 2 x 10/12.5 MVA, 33/6.6 kV transformers	17385	66063	Nil	Immediate
	ii) 6.6 kV/433 V distribution transformers	36288	137895	Nil	Immediate
	Total	53673	203958	-	-

4.0 ELECTRIC MOTORS

4.1 FACILITY DESCRIPTION

Electric motors, utilise a significant part of the total energy consumption of the plant. Electric motors are used to operate various equipments like pumps, fans, blowers, compressors, hammer mills, etc. There are 8 Nos. of HT motors operating on 6.6 kV

4.2 OBSERVATION, ANALYSIS AND FINDINGS

A. General

i. The motors of various horse powers used for various applications were analysed for loading power factor and efficiency related aspects. The measured parameters are tabulated in Appendix - 4/1.

ii. The power factor of motors which are adequately loaded are found to be satisfactory.

B. Operating Efficiency

The operating efficiency of motors rated below 22 kW have been estimated using an empirical formula and tabulated in Appendix - 4/2. The range of operating efficiency of different capacities is given in the table below :

Rated kW	Rated FL efficiency	Range of operating efficiency
22	89	50-89
18.5	89	75-89
15	88	48-88
11	88	76-88
7.5	85	55-85
5.5	85	61-85

C. High Efficiency Motors

It is to be noted here that many motors are more than 15 years old. Many of these motors have been rewound more than three times over their lifetime. Thus these motors will have low operating efficiency and many a time low operating power factor which is an indication of increase of no-load losses. Also the motors recommended for replacement have lower operating power factor than the motors of the same capacity operating at a similar load in the plant.

These motors can be replaced by energy efficient motors. The detailed calculation of energy savings that can be envisaged are given in Appendix - 4/3.

D. Power Factor Improvement

The operating power factor of individual motors is low and average operating power factor of different sections are listed in Appendix - 4/4. The average operating power factor varies from 0.34 to 0.82. It can be improved to 0.85. The following table gives the summary of operating power factors at different MCC.

Power factor range	No.of MCC
0.3-0.5	6
0.51-0.7	11
>0.71	3

The benefits of improving the power factor to 0.85 are:

i. Reduction in total current drawn from the mains resulting in gain in system capacity.

ii. Reduction in voltage drop enabling better performance of electrical equipment in the installation.

However, due to lack of space, additional building extension are required for installing capacitor banks which is not economical. Since p.f. has already been improvedc to 0.95 and above at 6.6 kV system (MRS), benefits of kVA demand savings have been already realised. It is proposed to instal 2 x 200 kVAr banks (presently kept switched off in acid plan s/s) provided the capacitor banks one in good condition ; These capacitor units may be installed at MCC's having low p.f. for minimising distribution losses.

E. Loading of Motors

The loading pattern of motors was studied. It is found that more than 50% of the motors are loaded below 50% of their rated capacity. Very few motors are loaded beyond their capacity.

The following table gives the summary of number of motors analysed plant-wise alongwith the percentage load variation.

Plant	No.of motors	% load variation
Roaster	19	7.7-81.0
Mercury recovery	1	80.5
Acid (200 TPD	5	9.5-95.0
Blend yard	5	19.6-71.6
Pump house	7	37.6-98.16
Cell house	22	3.27-115.91
Leaching	32	10.50-71.40
SFD	19	28.97-81.27
Cadmium	3	10.0-96.0
Charge preparation	5	17.33-93.60
DL plant	4	7 04-78.41
Crusher house	7	12.49-64.0
Gas cleaning	5	30.81-50.43
New blast furnace	9	5.73-57.82
Cooling tower	5	44.8-100.8
Lead refinery	8	9.55-64.86
Effluent treatment	11	26.73-74.18

The underloading of several of these motors is due to the kind of application they are used, like in agitators, feeders, conveyors, etc.

F. Optimum Sizing of Grossly Underloaded Motors

Motors operating below 40% of its rated capacity can be operated in the star mode. The rated output is reduced to one-third its rated capacity. The operating efficiency in the star mode is higher when the motor is operated in delta. However, the acidic and non-acidic fan motors of rated capacity of 45 kW are grossly underloaded. These motors are loaded to about 13.5% and 15.9% of its rated capacity respectively. These motors can be sized to 22 kW provided the starting torque requirement are met with. This measure also improves the power factor of load to above 0.7 lag. Detailed calculation of energy savings that can be envisaged is given in Appendix - 4/6.

G. H T Motors

i. The loading of the HT motors are quite satisfactory except the baghouse blower which is loaded only upto 34% of its rated capacity.

ii. The operating power factor of the adequately loaded motors is good.

H. Rewound Motors

There is a need to maintain history cards of motors with details of motor specifications and rewinding details such as number of times the motor is rewound, cost of rewinding, frequency of rewinding, etc. This should also include the no-load current and no-load power factor of the rewound motor. This information will help in assessing the suitability of the motor for a given application and also for justifying the replacement of motors.

4.3 RECOMMENDATIONS

A. Replacement of Standard Induction Motors by High Efficiency Motors

The identified motors should be replaced by high efficiency motors. Detailed calculations are given in Appendix - 4/3.

Energy savings	= 72330 kWh/year
Cost savings	= Rs.253155/-
Cost of implementation	= Rs.724000/-
Simple payback period	= 2.63 years

B. Optimum Sizing of Underloaded Motors

The acidic and non-acidic fan motors of Roaster plants can be run in the star mode as the motors are loaded only upto 13.5% and 15.9% of their rated capacity respectively. It is proposed to undersize the motor to 22 kW rating for optimum loading.

Energy savings	= 8640 kWh/year
Cost savings	= Rs.21940/-
Cost of implementation	= Nil
Simple payback period	= Immediate

C. History Cards for Motors

History cards of all motors should be maintained. This will enable the plant personnel in assessing the suitability of a motor for a given application and take decision regarding replacement of motors.

4.4 SUMMARY OF POTENTIAL SAVINGS

Sl. No.	Recommendation	Energy Savings kWh/yr	Cost savings Rs. lakhs	Investment required Rs lakhs	Payback period (yr)
1.	Replacing std. induction motors by high efficiency motors	72330	2.5[illegible]	7.24	[illegible]
2.	Operating grossly underloaded motors in star mode	8640	0.22	Nil	Immediate
Total		80970	2.75	7.24	2.63

* Savings in demand

5.0 STEAM GENERATION

5.1 FACILITY DESCRIPTION

The steam requirement of the plant is met by the waste heat recovery boiler in the Roaster plant and a standby Auxiliary boiler. Auxiliary boiler meets the total plants requirements during the Roaster plant shutdown.

The waste heat boiler, has steam generation capacity of 10.5 T/hr at 42.0 kg/cm^2g using the hot exhaust gases from Roaster at 900 °C as heating medium. The auxiliary boiler is of package type with steam generation capacity of 10 T/hr at 10 kg/cm^2g. Detailed specification of Waste heat boiler and Auxiliary boiler are given in Appendix - 5/1.

5.2 ENERGY CONSUMPTION

LDO consumption and running hours of the Auxiliary boiler for the year 1994-95 is given in Appendix - 5/2. The average hourly LDO consumption from the past year data is around 680 l/hr with total LDO consumption of 1290.5 kL.

5.3 OBSERVATIONS, ANALYSIS AND FINDINGS

5.3.1 AUXILIARY BOILER

A. Auxiliary Boiler Efficiency Calculation

Efficiency evaluation of Auxiliary boiler has been worked out by indirect heat loss method. The average values of various parameters observed for efficiency evaluation of the boiler is given below.

Sl. No.	Parameter	Unit	Value
1.	Trial duration	Hrs	3.0
2.	Average steam pressure	kg/cm^2g	4.5
3.	Average feed water inlet temperature	°C	30
4.	Average exit flue gas temperature	°C	172
5.	Average percentage of CO_2 in exhaust gas	%	9.5
6.	Average ambient temperature	°C	32
7.	Average air flow rate	m/s	23.85
8.	Average oil flow rate	kg/hr	432
9.	Gross calorific value of LDO	kcal/kg	10800

The calculations for thermal efficiency of the boiler has been given in Appendix - 5/3.

The efficiency of the boiler has been worked out to be 84.88 %.

The summary of heat losses and efficiency of the Auxiliary boiler are tabulated below :

SUMMARY OF HEAT LOSSES & EFFICIENCY

Sl. No	Heat Input/Heat Loss	KJ/kg	% Loss
1	Heat Input	10800	100
2.	Heat loss due to sensible heat in flue gases	773.70	7.16
3	Heat loss due to hydrogen in fuel ..	792.14	7.33
4	Heat loss due to moisture in air	29.72	0.247
5	Heat loss due to Radiation & Convection	-	0.347
6	Total losses	-	15.11
7	Thermal Efficiency	-	84.88

From the figures in the above table it can be observed that performance of the Auxiliary boiler is quite satisfactory.

B. Fuel Substitution

There exists a good potential to substitute the LDO being in use by furnace oil. Implementation of the above measure needs the replacement of the existing burner and providing preheating facilities for the day-service tank and fuel oil lines.

Implementation of the above measure is expected to yield annual savings of around Rs.9.5 lakhs with a simple payback of 2.6 years. Details are given in Appendix - 5/4.

C. Surplus Steam Generation

The steam of 34.0 kg/cm^{2}g is being reduced to 10.0 kg/cm^{2}g through PRV. There exists a potential for extra steam generation through addition of water in the process of pressure reduction. The quantification of extra steam generation has been given in Appendix - 5/5. Though potential exists for extra steam generation, in view of already surplus steam is available in the plant this proposal is not worth examining in detail.

D. Estimation of Degree of Superheat Steam

Steam at a pressure of 10.0 kg/cm^{2}g and 34.0 kg/cm^{2}g are at saturation temperature of 183.2 °C and of 241.42 °C respectively. Through pressure reduction the saturated steam gets superheated. The details and estimation of the degree of superheat achievable is given in Appendix - 5/6. Degree of superheated steam achieved is 195.54 °C.

5.4 RECOMMENDATIONS

A. In the present working of boiler LDO is being used as fuel. It should be substituted by Furnace Oil. Implementation of this measure is expected to result in annual savings of Rs.9.5 lakhs with a simple payback period of 2.6 year. Refer Chapter 5.3.1 B for details.

Energy savings	=	130 kL LDO
Savings in Rupees	=	9.5 lakhs
Cost of implementation	=	25.00 lakhs
Simple payback period	=	2.6 years

5.5 SUMMARY OF POTENTIAL SAVINGS

Sl No.	Proposal	Estimated Energy Savings		Cost savings Rs.lakhs	Cost of implementation Rs lakhs	Simple payback period (years)
		Thermal kL/yr	Electrical (kWh/yr)			
1	Fuel substitution	130	-	9.5	25.00	2 6
	Total	130	-	9.5	25.00	2.6

6.0 STEAM DISTRIBUTION & UTILISATION

6.1 FACILITY DESCRIPTION

The generated steam at a pressure of 34.0 kg/cm² is reduced to a pressure at 10.0 kg/cm²g through pressure reducing valve. Steam at a pressure of 10.0 kg/cm²g is suffice to meet the various user areas requirements.

The major steam utilisation areas in the plant are :

i. Leaching & Purification plant
ii. Tail Gas Treatment plant

6.2 OBSERVATIONS, ANALYSIS & FINDINGS

A. Insulation Aspects

A survey of steam distribution network was carried out to identify and quantify the heat losses from uninsulated pipes, flanges, valves and equipments. The total heat losses from uninsulated steam surfaces are given in Appendix - 6/1. The total heat loss has been estimated to be around 7287.04 kcal/hr. The estimated heat loss after insulation is around 794.63 kcal/hr. Possible energy savings by insulation is approximately 1.565 kL of LDO (i.e, Rs.11,440). At an estimated investment of Rs.18,000/- it works out to a simple payback of 1.6 years.

B. Steam Leakages

Observed steam leakages from the glands and pipes and quantification of the same is given in Appendix - 6/2. It works out to be approximately 45.6 kg/hr of steam. The annual energy savings by plugging steam leakages works out to approximately 7.36 kL of LDO annually (i.e, Rs.63290/-). Plugging of steam leakages at an estimated investment of Rs.25,000/- works out to a simple payback period of 0.40 years.

C. Trapping System

Traps provided for steam main are of thermodynamic type. A survey of trapping system reveals 5 nos. of thermodynamic traps in the steam line to leaching plant are not functioning.

D. Steam Utilisation Pattern

Steam is used indirectly in heat exchangers directly in pachuka's in Leaching & Purification plant and in Tail Gas Treatment Plant. The steam pressure requirement of Leaching & Purification plant is 6-8 kg/cm^{2}g & 3-4 kg/cm^{2}g for Tail Gas Treatment plant. The steam requirement at full load for (i) Tail Gas Treatment plant & (ii) Leaching & Purification plant have been worked out and given below :

Sl. No.	Area	Steam Pressure kg/cm^{2}g	Steam quantity kg/hr
1.	Tail Gas Treatment	3.5	155
2.	Leaching & Purification	7.0-8.0	14300*

* For maximum of 4 Pachuka operation

Calculation details are discussed in Chapter 13.

6.3 RECOMMENDATIONS

A. Insulation of uninsulated steam mains, flanges and valves should be taken up. This has the potential annual energy savings to the tune of 1.565 kL LDO (i.e, Rs.11440/-) with a simple payback of 1.6 years. Refer chapter 6.2 A for details.

Energy savings	=	1.565 kL /Annum
Savings in Rupees	=	Rs.11440/-
Cost of implementation	=	Rs.18,000/-
Simple payback period	=	1.6 years

B. Existing steam leakages as identified should be plugged at the earliest. Annual energy savings to the tune of 7.36 kL of LDO (i.e, Rs.63290/-) exists. At an estimated investment of Rs.25,000/- works out to a simple payback period of 0.4 years.

Energy savings	=	7.36 kL/Annum
Savings in Rupees	=	Rs.63290
Cost of implementation	=	Rs.25,000/-
Simple payback period	=	0.4 years

6.4 SUMMARY OF POTENTIAL SAVINGS

Sl No	Proposal	Estimated Energy Savings		Cost savings Rs. lakhs	Cost of implemen -tation Rs lakhs	Simple payback period (years)
		Thermal kL LDO/yr	Electrical (kWh/yr)			
1	Insulation of uninsulated pipe fittings	1 565	-	0 1144	0 18	1 6
2	Plugging of steam leakages	7 360	-	0 6329	0 25	0 4
Total		8 925	-	0 7473	0 43	-

7.0 CO-GENERATION

7.1 FACILITY DESCRIPTION

In the existing system, Roaster gases from the furnace at a temperature of 900 - 950 °C enter the Waste Heat Boiler which is designed to produce steam of 10.5 MT/h at 42 kg/cm^{2}g. Designed outlet gas temperature is expected to be around 360 °C.

7.2 OBSERVATIONS, ANALYSIS AND FINDINGS

A. Power Recovery Potential

An assessment of heat recovery potential has been worked out for estimated steam generation of 13,500 kg/h at a input steam pressure of 37 kg/cm^{2}a (Super-heated to 60 °C) and a back pressure of 4 ata (Super-heated to 15°C). The power recovery potential is estimated to be 293 kW. (Calculation details are given in Appendix - 7/1). Though, around 293 kW power recovery potential exists, practical considerations do not permit as the exit gas temperature from waste heat boiler is around 350 °C - 360 °C only.

Dew point considerations and requirement of high gas temperature for production of super-heated steam limit its practicable viability. Use of external fuel is also ruled out as it calls for major design modification.

8.0 COMPRESSED AIR SYSTEM

8.1 FACILITY DESCRIPTION

The details of Make, number of Compressors, Type, Rated FAD capacity, Rated kW are as given below :

Sl. No.	Compressor	Make	Nos.	Type	FAD capacity (m^3/hr)	Rated kW
1.	Process & Instrument Air Compressor	Ingersoll-Rand	1	Centrifugal	4549.65	522
2.	Process Air Compressor	Khosla-Crepelle	4	Reciprocating	1698	248
3	Instrument Air Compressor	Khosla-Crepelle	2	Reciprocating	420.0	55.8
		Kirloskar	[illegible]	Reciprocating	522.5	75.0
		Kirloskar	[illegible]	Reciprocating	[illegible]	75.0

Normally centrifugal compressor meets the requirement of both instrument and process air. During study centrifugal compressor was in operation and others were kept as standby.

The compressor specifications along with design values are given in Appendix - 8/1. The compressor is provided with heaterless type drier unit of 50 m^3/min capacity to deliver moisture free air to all user areas.

The major compressed air consumption areas is as given below :

PROCESS AIR USER AREAS :

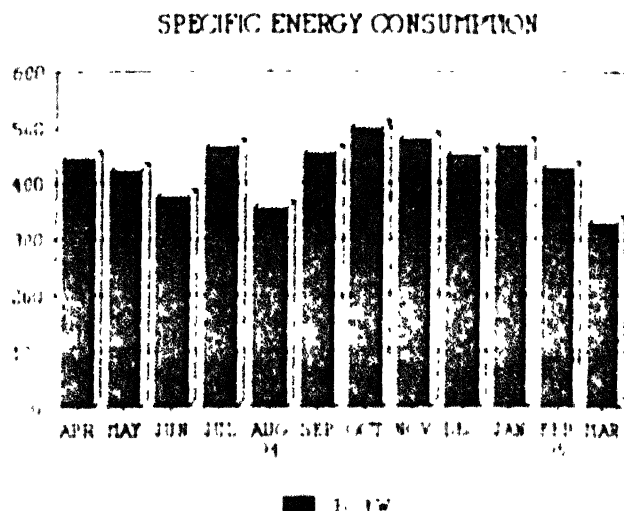
Sl. No.	Usage	Purpose	Operation
A.	ROASTER PLANT		
1.	Under flow conveyor	Cleaning	Intermittent
2.	Roaster	Lancing	Intermittent
3.	Roaster chute	Choke removal	Intermittent
4.	Waste heat boiler	Furnace bundle cleaning	Intermittent
5.	Redler conveyor No.8	Choke removal	Occasionally
B.	**SULPHURIC ACID PLANT 200 TPD**		
1.	**Preheater burner**	**Atomisation air when it runs**	**Continuous**

INSTRUMENT AIR USER AREAS :

Sl No.	Usage	Purpose	Operation
A.	ROASTER PLANT		
1.	Steam drum	Inlet valves Outlet valves	Continuous Continuous
2.	Deaerator	Steam valves	Continuous
		Vent valves	Continuous
B.	SULPHURIC ACID PLANT 200 TPD		
1.	Circulating tanks	Absorption tower & Drying tower (2 Nos.)	Continuous
2.	SO_2 blower	Vane control valves	Continuous
C.	SULPHURIC ACID PLANT 50 TPD		
1.	Preheater	Fuel oil control valve (1 No.)	
D.	TAIL GAS TREATMENT PLANT		
1.	Circulating tank	Level indicators (2 Nos.)	Continuous
E.	LEACHING AND PURIFICATION		
1.	Pachuka's	Agitation	Occasional when power fails
2.	Slime dorr thickener	Pneumatic transfer of under flow	Continuous

8.2 ENERGY CONSUMPTION PATTERN

The 720 HP centrifugal compressor has separate energy meter. The annual energy consumption and running hours for the year 1994-95 for Centrifugal Compressor is around 29.73 lakh kWh and 6812 hours respectively.

The average energy consumption of the compressor is about 436.48 kWh. Month-wise energy consumption and running hours for the year 1994-95 for Centac Compressor and running hours for the year 1992-93 to 1994-95 of Process Air compressors & Instrument Air compressors are given in Appendix - 8/2.

8.3 OBSERVATIONS, ANALYSIS & FINDINGS

A. Performance Assessment of Centrifugal Compressor

The centrifugal compressor was studied to evaluate its performance. The hourly energy consumption of the compressor on the day of study has varied from 315-495 kWh. The loading on the compressor motor is around 60.34% - 94% of its rated capacity. Observed temperature and pressure values of intercoolers and aftercoolers are given in Appendix - 8/3

An average temperature rise of 10.2 & 11.1 °C across the I stage intercooler and II stage intercooler respectively indicates satisfactory performance of intercoolers.

B. Performance Assessment of Process Air Compressors

Performance evaluation of the process air compressors could not be evaluated. They were kept as standby. Attempt has been made to look to their performance from the past data recorded and is given in Appendix - 8/5. The summary of the observations are given below :

Data	PAC-I	PAC-II	PAC-III
Average oil pressure (kg/cm^2g)	2 80	2 3	2 4
LP 'A' Pressure (kg/cm^2g)	1 50	1 56	1 8
LP 'B' Pressure (kg/cm^2g)	2 3	1 40	1 5
After cooler pressure (kg/cm^2g)	5 0	5 0	-
Receiver pressure (kg/cm^2g)	4 9	4 9	-
Outlet water temperature (°C)			
- Intercooler	31 33	-	-
- Aftercooler	44 00	-	-
Final air temperature (°C)	115 0	-	-
Current (Amps)	242 0	205 0	223 1
Average LP Pressure	1 90	1 48	1 65
Drop in air temperature after intercooler (°C)			
'A' Block	26 0	31 7	24 0
'B' Block	2 0	-	-

From the above table it can be observed that the figures of outlet air temperatures after the aftercoolers is on the higher side, indicating the poor performance of aftercoolers which will lead to lower free air delivery than the rated FAD.

C. Cooling Water Circuit

The temperatures of water from intercoolers and aftercoolers from Centac compressor was around 34 - 37 °C . This water is being pumped to Lead Refinery cooling tower after collecting them in a ground tank. The temperature of the water in ground tank is around 35 - 39 °C due to a hot stream joining from Leaching plant. The above rise in water temperature increases the total cooling load on the cooling tower. Additionally, it was observed that there was continuous overflow of water from tank to effluent. Supply of this water affects the efficiency of Lead Refinery cooling tower.

D. Installation of Separate Cooling Tower

The continuous overflow of water from the ground water tank to the effluent plant has increased the water demand of the compressor house. In normal practice, compressor house is provided with separate cooling towers, in order to reduce the water demand. Hence it is very much envisaged to install a new cooling tower.

E. Free Air Delivery Test (FAD)

The test could not be carried out due to continuous process requirement of instrument air. However, this must be carried out at regular intervals to assess the free air delivery capacity of the compressor and its specific energy consumption. Details of the tests are given in Appendix 8/6.

9.0 WATER PUMPING & COOLING TOWERS

9.1 ZINC PLANT

9.1.1 FACILITY DESCRIPTION

A. WATER PUMPING

The plant has its own water reservoir to meet its total maximum water demand of 3.5 million gallons per day. Water to plant is pumped through 75 kW motor. The average consumption varies between 7000 - 8000 m^3/day. The major user areas of water with break-up is as given below.

Sl. No.	Plant/Section	% of Total requirement	Average Water consumption m^3/day
1	Roaster plant	9	630 - 720
2	Acid plant (200 TPD & 50 TPD)	3	210 - 240
3.	Zinc Leaching & Purification	4	280 - 320
4.	Zinc Electrolysis & Rectifying	6	420 - 480
5.	Melting section	3	210 - 240
6.	Cadmium plant	1	70 - 80
7.	Silver Flotation	8	560 - 640
8.	Compressor House	12	840 - 960
9.	Effluent Treatment plant	3	210 - 240
10.	Zinc Oxide plant	1	70 - 80
11.	Lead plant	29	2030 - 2320
12.	Drinking (Township & Plant)	21	1470 - 1680
	Total	100	7000 - 8000

B. COOLING TOWERS - ZINC PLANT

i. ROASTER & ACID PLANT

The plant has Three Nos. of cooling towers to cater to cooling water requirement of Calcine cooler, Roaster cooling & Plate heat exchangers of acid plant. The specifications of cooling towers is given in Appendix - 9/1. The details of cooling towers and their application are as given below :

Cooling Tower	Type	No. of cells	Application Areas
Roaster & Acid Plant	Induced draft cross flow	2	1.Acidic Areas a.Plate heat exchangers in acid plant b.Cooling of SO_2 blower c.Acid Mixing d.Air conditioning & Blend yard 2.Non-Acidic Areas a.Stand pipe b.Scrubber c.Star cooler Cooling Tower No.1
Calcine Cooling	Induced draft cross flow	1	1.Calcine conveyor system a.Overflow & underflow of waste heat boiler b.Cyclone separator
50 TPD Acid plant	Induced draft cross flow	1	50 TPD Acid plant

9.1.2 OBSERVATIONS, ANALYSIS & FINDINGS

A. Cooling Tower Pumps

Loading pattern of cooling tower pump motors are given below.

Cooling Tower	Pump No	Rated kW	Actual kW	% Loading
Roaster & Acid plant	1 2 3	110 110 110	50.4 85.2 65.4	45.8 77.5 59.5
Calcine cooling water	1 (Hotwell) 2 (Coldwell)	30 18.5	21.15 12.90	70.5 69.7

From the table it can be observed that the loading on the motors are satisfactory.

B. Cooling Tower Fans

Loading pattern of cooling tower fan motors are given below.

Cooling Tower Area	Fan No.	Rated kW	Actual kW	% Loading
Roaster& Acid plant	1 2	45 45	6.09 7.17	13.5 15.9
Calcine cooling water	1	7.5	3.60	48.0

From the table it can be observed that the motors of Roaster plant are very much underloaded. Details regarding conversion to star mode operation are discussed in chapter 3.

C. Performance Evaluation of Cooling Towers

Towards performance evaluation of cooling towers of Roaster & Acid plant and calcine cooling the following parameters were monitored.

i. Cooling water inlet & outlet temperatures
ii. Ambient dry & wet bulb temperature

From the above measured values, range & approach have been computed for all the cooling towers. The details are given in Appendix 9.1/2.

The highlights of the above observation are given below :

SUMMARY OF COOLING TOWER PERFORMANCE (AVERAGE)

Cooling Tower No	Cell No	Average outlet water temp (°C)	Dry Bulb temp (°C)	Wet Bulb Temp. (°C)	Approach (°C)	Range (°C)
Roaster & Acid Plant	1 2	34 4 34 0	33.0 33.0	27.60 27 60	6.8 6.4	8.8 9 2
Calcine cooling	1	27.5	30.80	26.70	0.75	2.5

The range and approach values in the table indicate satisfactory performance of cooling towers.

D. Cooling Water Analysis

Roaster & Acid Plant cooling tower water samples were tested for pH, TDS & TSS. Observation of the values shows the TDS levels are within the norms.

9.2 LEAD PLANT

9.2.1 FACILITY DESCRIPTION

Lead plant is provided with three cooling towers viz., two cooling towers in sinter plant and one cooling tower in lead refinery plant.

The details of cooling towers and their application areas are tabulated below :

Location	Type	No.of cooling towers	Application area
Sinter plant	Induced draft cross flow	2	Drum mixers, gas cleaning plant, blast furnace, slag settler, etc
Lead refinery	Induced draft counter flow	1	HT motor bearing, vacuum dezincing, casting machine, etc

9.2.2 OBSERVATIONS, ANALYSIS AND FINDINGS

A. Requirement of Cooling Water in Lead Smelter Plant

The cooling water requirement in smelter section along with pressure required are tabulated below. Equipment-wise water requirements are given in Appendix - 9.2/1.

Section	Pr.reqd. kg/cm^2g	Water reqt. L/h
Charge preparation	1	500
Sinter machine	4	3000
Blast furnace	2	153340
Crusher section	4	1100
Gas cleaning	2.5	-

B. Performance Evaluation

The performance evaluation of cooling towers in smelter plant and lead refinery were carried out. The parameters monitored were inlet, outlet temperatures of cooling water, dry and wet bulb temperatures. The tabulated values are given in Appendix - 9.2/2. The range, approach and efficiency of cooling towers are tabulated below:

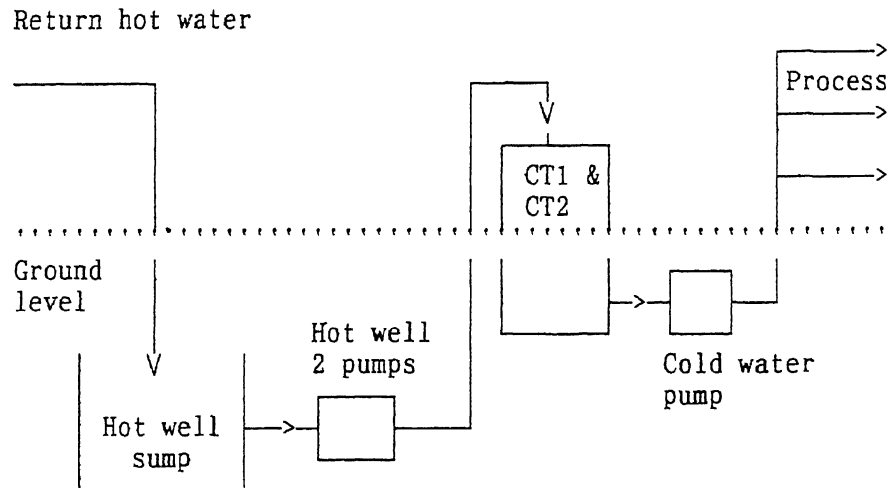
Sl No	CT-1			CT-2			Lead refinery		
	R	A	E	R	A	E	R	A	E
1	2	7	28.5	1.0	10.0	10.0	2.0	3.5	57.14
2	4	9	44	1.5	14.0	10.7	2.0	3.0	66.67
3	4	9.5	42	1.0	13.5	7.4	2.5	3.5	77.42
4	5	10.0	50	1.5	14.5	10.34	3.5	5.5	63.63

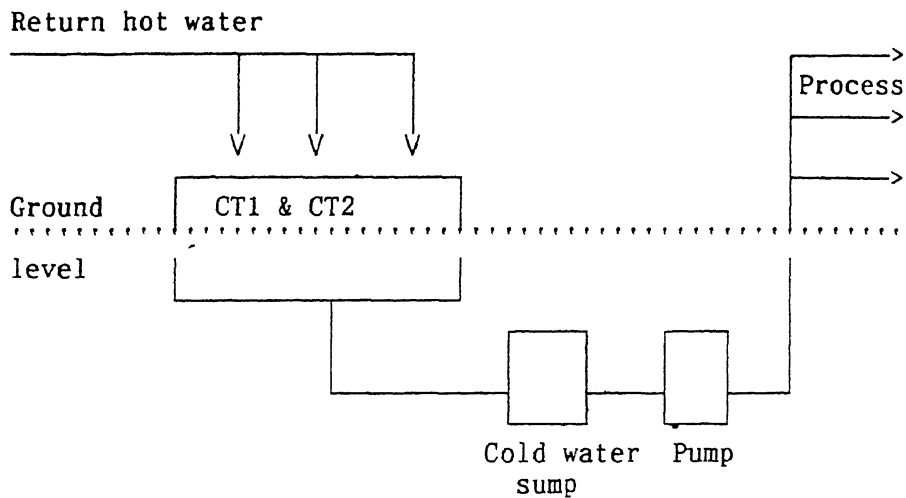
The cooling towers in lead smelters are operating at very low efficiency, while refinery cooling tower is operating at moderate efficiency.

The badly damaged cooling tower wooden fills, absence of forced air circulation are resulting in low efficiency of cooling towers . These cooling towers should be reconditioned to increase the efficiency inturn to reduce the outlet water temperature. Low outlet temperature of water will also improve the process parameters at user ends.

C. Application of Once Through Cooling System

The cooling water supply in smelter plant is serviced by hot well and cold well pumps. The water meeting the various process cooling needs (except in plate heat exchanger and drum mixers) is pumped by two pumps to cooling towers from the hot well sump. The outlet cold water of cooling tower is pumped to various section of the plant by a cold water pump.

Present System

Since the return water has sufficient head due to elevation, the return line can be directly connected to the top of the cooling tower. The existing hot well sump can be used as cold well sump.

Proposed system

Application of once through system avoids two hot well pumps in the circuit. To enable free gravity flow the existing return line has to be replaced with 14' pipe. Separate strcutural support is required to lay this pipe line. IN order to avoid the overflow of water during power failure period from hot sump and cold sump, addition of 140 m^3 capacity is required. The annual energy savings which would accrue by implementation of the measure is expected to yield annual energy savings of 1.408 kWh ie., Rs.5.35 lakh, with an initial investment of Rs.8.00 lakh. The details are given in Appendix - 9.2/3.

9.2.3 RECOMMENDATIONS

Once through Cooling Water Pumping System

It is preferable to have once through pumping system eliminating hot well pumps. Envisaged investment is expected to payback in less than one year. Refer Section 9.2.2 C for details.

Estimated savings = 1.408 lakh kWh/yr
Cost savings = Rs.5.35 lakh/year
Investment required = Rs.8.00 lakhs
Simple payback period = 1.5 year

9.2.4 SUMMARY OF POTENTIAL SAVINGS

Recommendation	Energy Savings (L.kWh/yr)	Cost savings Rs.lakhs	Investment required Rs.lakhs	Payback period (yrs)
Once through cooling water pumping system	1.408	5.35	8.00	1.50
Total	1.408	5.35	8.00	1.50

9.3 COOLING TOWER - D.G. POWER HOUSE

9.3.1 FACILITY DESCRIPTION

The plant has one cooling tower to cater to cooling water requirements of DG set oil cooler, & DM water cooling. The cooling tower is of 600 NM^3/hr capacity. The detailed specifications of the cooling tower is given in Appendix 9.3/1.

9.3.2 OBSERVATIONS, ANALYSIS & FINDINGS

A. Cooling Tower Fans

Loading pattern of cooling tower fan motors are as given below :

Fan No.	Rated kW	Actual kW	% Loading
1	24.1	21.1	92.1
2	24.1	19.5	87.5

It can be observed that loading is satisfactory.

B. Performance evaluation of cooling towers

Towards performance evaluation of cooling towers, the following parameters were monitored.

i. Cooling water inlet & outlet temperatures
ii. Ambient dry & wet bulb temperatures

From the above measured Values, Range, Approach have been computed and are given in Appendix 9.3./1B. The highlights of the above observation are given below :

Average outlet water temp. °C	Dry bulb temp. °C	Wet bulb temp. °C	Approach °C	Range °C
32.3	28.9	26.8	5.5	16.1

From the above observations, it can be concluded that the performance of cooling tower is satisfactory.

C. Use of Soft Water

Raw water is in usage for the cooling of oil coolers & DM water in DG sets. Usage of raw water in the cooling circuit, increases the rate of scale formation, & corrosion, thereby decreasing the heat transfer rate across the surface. Better performance and reduced down time can be achieved by use of soft water. The existing water softening unit in the plant is not in operation due to operational problems. Hence it is advisable to use soft water in DG Sets.

D. Water Distribution in Cooling Tower

The water distribution pattern in the cooling tower has been found to be not even much overloading of one cell keeping the other empty. Even distribution of water increases the efficiency of cooling tower & reduces the overloading of fan.

10.0 PUMPS, FANS AND BLOWERS

10.1 FACILITY DESCRIPTION

Plant uses Pumps, fans and blowers for handling various process liquids such as spent electrolyte, acids, process and cooling water, combustion air and dust removal system.

10.2 OBSERVATIONS, ANALYSIS AND FINDINGS

Loading Pattern of Pumps and Blowers

The observed plant-wise loading pattern of various pumps, fans and blowers have been tabulated in Appendix - 10/1. Though loading pattern suggest satisfactory operating condition of various equipments, the following pumps have been identified for further detailed examination. These pumps have loading pattern much below 50%. In view of non availability of design flow and head, no positive conclusion could be drawn. Once these parameters are available, liquid horse power should be worked out using the relation :

$$\text{Liquid Horse Power} = \frac{\text{Flow (gpm)} \times \text{Head ft}}{3960}$$

Considering an overall pump-motor efficiency of 0.5, required drive power can be arrived. Pumps having loading pattern less than 50% are detailed below :

Sl. No.	Application	Rated kW	Actual kW	% Loading
	PUMP HOUSE			
1.	Filter water pump	75	33.39	44.5
2.	Emergency water pump	110	52.5	47.7
	ROASTER PLANT			
1.	Process water pump 3	110	50.4	45.8
	CELL HOUSE			
1.	Electrolyte pump 47	37	3.63	9.68
	LEACHING PLANT			
1.	ZnO_2 Ball mill pump 32	18.5	6.93	37.46
2.	ZnO_2 Ball mill pump 23	18.5	5.82	31.46
3.	Pachuka discharge pump 07	15	5.85	39.00
4.	New pachuka discharge pump	11	3.3	30.30
	SILVER FLOTATION DEPT.			
1.	Neutralisation tailing pump	15	4.74	31.60
2.	Agitator lime slurry pump 14	11	3.51	31.91
3.	Rectifier return pump	11	3.75	34.08
	GAS CLEANING PLANT			
1.	Hot water sump pump 2	22	8.58	39.00
2.	RC pump 1 B	18.5	6.06	32.76
3.	Stripper feed pump 22 A	18.5	5.7	30.81
	NEW BLAST FURNACE			
1.	Scrubber pump 3	18.5	3.75	20.27
	EFFLUENT TREATMENT PLANT			
1.	Horizontal pump 1	11	3.9	35.45
2.	Lime agitator pump 1	5.5	1.95	35.45
3.	F D Pump	30	8.73	29.10
4.	Air Blower	15	5.82	38.80

Note : A few of the low loaded pumps and blowers have not been included above considering varying load operation.

11.0 ROASTER PLANT

11.1 FACILITY DESCRIPTION

The Zinc concentrates known as Zinc blende, ZnS, Containing about 48% to 55 % Zn and 30% Sulphur are roasted in a fluo solid roaster to convert ZnS into its oxide form which is easily soluble in dil sulphuric acid. The Roaster plant and Gas Cleaning section consists of : i. Concentrate Handling System ii. Furnace iii. Calcine Handling System iv. Waste Heat Boiler v. Hot Gas Cleaning Section vi. Wet Gas Cleaning Section. Main features of the fluo-solid roaster designed by Lurgi Chemec, Germany is given in Appendix - 11/1. Fluidisation air is met from a centrifugal type blower whose maximum capacity is 25000 Nm^3/hr at 2300 mm wc. LDO is employed for initial start up as well as for lancers provided for raising the temperature upto 800 - 880 °C. Roaster zinc concentrate which is known as calcine has distribution of calcine generated as follows :

Furnace overflow / underflow	= 30 %
Waste Heat Boiler	= 30 %
Cyclones	= 37 %
Hot gas precipitator	= 3 %

Waste Heat Boiler

Dust laden gases from the furnace at a temperature of 900 - 950 °C enter the waste heat boiler which is designed to produce steam of 10.5 MT/hr at 42 kg/cm^2. Salient features are given in Appendix - 11/1.

11.2 ENERGY CONSUMPTION

Roaster plant consumes LDO for preheating during start up and maintaining temperatures at the Roaster bed. The annual LDO consumption for 1994-95 works out to 121 kL. Monthwise Roaster running hours and preheating burning hours of burners and lancers are given in Appendix - 11/2.

11.3 OBSERVATIONS, ANALYSIS AND FINDINGS

A. Surface Heat Losses

Roaster surface is provided with fire clay brick, insulation brick and hysil insulation. Towards assessment of adequacy of insulation provided and quantification of heat losses, surface temperature measurements were carried out sectionalising the roaster into different zones. Observed average surface temperature and quantified heat losses are worked out in Appendix - 11/3. Encountered average surface temperature is in the range of 55 °C to 90 °C.

The summary of heat losses are given below :

Roaster section	Area (m^2)	Heat loss kcal/h	% of Total heat loss	% of Total area
B Section	77.600	38551.86	18.14	26.88
A Section	104.276	68745.70	32.36	36.12
C Section	106.80	105229.12	49.52	37.00
Total		212526.68	100	100.00

B. Observations on Process Parameters of Various Equipments

An exercise was undertaken to assess the performance of the various equipments employed by comparison of Design and actual temperatures and pressures of roaster gases. Design and observed temperatures and pressures of various process equipments are given in Appendix - 11/4 and 11/5. Observation highlights are given below :

Sl No	Area/Equipment	Temperature Drop (°C)		Pressure (mmwg)	
		Design	Actual	Design	Actual
1	Furnace	868.915	933	1700	1700
2.	Boiler bundle (1)	250-300	355	40	32
3	Cyclone separator	20	33	100	48
4.	Hot gas precipitator	0-30	25	30	20
5	Scrubber	233-263	239	30	20
6	Stand pipe	-	-	'90	170
7.	WGP - I	-	-	30	35
8.	Star cooling	-	-	100	80
9.	WGP -II	-	-	30	115

Minor variations of observed parameters in comparison to design values show the satisfactory operation of various equipments.

C. Waste Heat Boiler Surface Heat Loss Estimation

Waste heat boiler has been studied for quantification of the surface heat loss from its insulated surfaces. The details of average surface temperatures and heat loss are tabulated in Appendix 11/6. The percentage heat loss in different sections has been given below :

Sl. No.	Section/Area	Area m^2	Heat loss per m^2 kcal/hr/m^2	Heat loss kcal/hr	% Loss	% Area
1.	3 rd and 2nd floors (Both sides)	85 32	343.78	29331 20	57 98	59 76
2	Coil section (Both sides)	13 18	322 92	4256 14	8 40	9 23
3	Ist floor (Both sides)	44 28	329 44	14587.55	28 84	31 01
4.	Heat loss from duct	-	-	2408 74	4 78	-
	Total	-	-	50583.63	100.00	100.00

From the observed surface temperatures, the surface heat losses have been observed to be within practical permissible limits. Besides, observed boiler exit temperature of 356 °C vis-a-vis design value of 360 °C shows effective operation of waste heat boiler.

12.0 SULPHURIC ACID PLANT

12.1 FACILITY DESCRIPTION

The 200 TPD Acid plant is based on the process know-how from Ugine Kuhlmann of France. The important components/equipments in the acid plant are (i) Drying Tower (D.T), (ii) Blower, (iii) Converter, (iv) Heat Exchangers I, II, III & IV, (v) Absorption tower (A.T), (vi) Serpentine Coolers. The gases containing 5.5 % to 6.5% SO_2 and carrying 1200 kg/hr of moisture are dried in the D.T. The clean dry gases are drawn by a blower having 35000 Nm^3/hr capacity with 2950 mmwg outlet pressure and passed through two heat exchangers so as to attain a temperature of about 420 °C before entering first mass of the converter. For initial start-up of plant a pre-heater is employed to heat gases to 420 °C. (Typical design inlet and outlet temperatures of the four converter beds and other features are given in Appendix - 12/1).

12.2 ENERGY CONSUMPTION PROFILE

The total annual production of white and black acid (for 1994-95) together comes to 52037 MT which corresponds to capacity utilisation of around 69.4%. Month-wise production is given in Appendix - 12/2. The annual white and black acid production, plant running hours, preheater running hours and LDO consumption have been tabulated below. Month-wise details for 200 TPD and 50 TPD plants are given in Appendix - 12/3.

Year	Production MT	LDO Consumption kL	Running Hours	Pre-heater Running Hours
1994 - 95	49471.876 (White acid)	181.0	6043.65	668.85
	2565.21 (Black acid)	125.0	2372.9	2935.70
Total		306.0		

The total annual LDO consumption works out to 306 kL.

12.3 OBSERVATIONS, ANALYSIS AND FINDINGS

A. Preheater Efficiency Evaluation

Preheater provided in 200 TPD plant is utilised for preheating SO_2 gases during initial startup. Preheater is provided with both combustion and dilution air blowers. Preheater specification details are given in Appendix - 12/3. An efficiency trial carried out revealed the following values of various parameters monitored.

Sl. No.	Parameter	Units	Value
1.	Furnace temperature	°C	600
2.	Stack gas temperature	°C	360
3.	Combustion air pressure	mmwg	450
4.	Dilution air pressure	mmwg	280

Though observed stack temprature is 860 °C, plant personnel have reported the above value to be in the range of 260 - 270 °C. Hence the heat recovery prospects have not been considered.

B. Substitution of LDO by Fuel Oil

The potential to substitute the fuel being used in the initial preheating of the plant has been examined. The present LDO being used can be substituted by furnace oil, which is lower in cost. In view of lower running hours and frequent start and stop operation, the proposal is not techno-economically feasible.

13.0 LEACHING AND PURIFICATION

13.1 FACILITY DESCRIPTION

Calcine is leached with return spent acid in batches and the zinc sulphate solution produced is purified from impurities in a continuous process and pumped to Zinc Electrolysis plant. Leaching and purification plant consists of (a) pre-leaching (b) Neutral leaching (c) Slime leaching and (d) purification. The plant uses steam for both neutral and slime leaching. Various transfer pumps are the other energy consuming equipments.

13.2 OBSERVATION, ANALYSIS AND FINDINGS

A. Estimation of Steam Consumption

Both direct and indirect steam consumption has been estimated from first principles and given in Appendix - 13/1 and Appendix - 13/2 respectively. Maximum direct steam consumption considering four Pachuka's in heating condition at a time works out to 7760 kg/hr. Estimated indirect steam consumption is worked out to be 6540 kg/hr.

B. Surface Temperature Observations

Surface temperature observations of various equipments have been given below :

Sl No.	Equipment ref.	Electrolyte temp.°C
1	Neutral Pachuka's	42
2	Slime Pachuka	42
3	Neutral dorr thickner	41
4	Slime dorr thickner	42
5	Surge tank	41

All the above tanks are lined with acid resistant bricks. Maximum surface heat losses from each of the neutral pachuka's works out to 7000 kcal/hr.

Observed surface heat losses are well within the permissible limits.

C. Observations on Electrolyte Temperature

Equipment ref.	Surface temp.°C
Neutral Pachuka's	70
Neutral dorr thickner	62
Surge tank	57
Slime Pachuka	62
Slime dorr thickner	59

The above observations reveal a temperature drop of 8°C between neútral pachuka's and dorr thickner and 5°C drop between neutral dorr thickner and surge tank.

D. Evaporation Losses

A drop in temperature of 8°C is mainly due to evaporation losses from the surface. Total heat losses from the surface is estimated to be 611800 kcal/hr. However, during draft report discussion, the plant prsonnel reported that the temperature drop would not be more than 2 °C to 3 °C on a continuous basis. Also, there should be some evaporation loss allowed for maintaining water balance. Hence the prospect of retaining heat by thermocole sheets has not been considered.

14.0 ZINC ELECTROLYSIS PLANT

14.1 CELL HOUSE

Electrolysis plant mainly comprises of two separate rectifiers and cascade circuits X-12 and X-22 supplied by two power sources, pumping & cooling of electrolyte by 70 and 80 series pumps, and stripping of zinc deposited on the cathode. This is the major power load for the plant, consuming about 71.1 % of electricity.

14.1.FACILITY DESCRIPTION

Power for electrolysis is derived from two rectifier transformers of 9460 kVA rating, on two circuits. Supply voltage from rectifier room is at 580 Volts with a load of 13000 Amps supplied through copper busbars to these cascades. (Groups of cells). The two individual circuits (not in parallel) comprise 18 cascades each. Each cascade comprise 10 cells arranged in two rows of 5 each in series. (Details are given in the sketch 1 A enclosed). The DC voltage supply is not an earthed system.

Generally out of 18 cascades, one cascade is shutdown for maintenance, thereby 17 cascades ie., 170 cells are under electrolysis for production. All the cells are in series in both the circuits. Quick reference table is given below :

Particulars	Data
Rectifier circuits	X-12 & X-22
No. of cascades in each circuit	18
No. of cells in each cascade	10 nos. 5 on A side 5 on B side
Connection of cascades/cells	18 cascades in series ie., 180 cells in series
No. of Anodes in each cell	28
No. of cathodes in each cell	27
Current density A/m^2	400 - 430

The designed cell voltage is 3.3 Volts. Copper busbars over the cells have a wedge to accommodate copper tipped anodes and cathodes. The anodes and cathodes at the other side of copper bus section are supported by wooden (guide) supports with notches for getting a uniform gap of 65 mm between anodes and cathodes.

The anodes are 28 nos. per cell ; Material of construction is lead with a header and hook for manual lifting and cleaning.

The cathodes are 27 nos. per cell. Material of construction is aluminium with similar header and hook for manual lifting and stripping of zinc sheets after electrolysis.

The cell dimensions are 2700 mm x 1100 mm x 1550 mm with a holding volume of 2.8 m^3. The current density is 400 - 430 Amps per Sqm. The average zinc content in the electrolyte is 150 g/l. Cathode stripping cycle is 24 hours and this is done manually by chiseling action on the cathode plates.

Some of the optimum features of cells are given below : The average temperature of the cell feed (zinc sulphate solution) is 36-37 °C with normal evaporation of 1.1 %. The temperature of the cells will be at 43 to 44 °C.

The main FRP launder carrying the zinc sulphate solution is fed by three or four 80 series pumps operating to give a total flow rate of 600 gal/min. The cell feed is by gravity to individual cascade launders. Neutral electrolyte is pumped at the rate of 100 gal/min to the main launder.

The spent electrolyte from each cell is collected in common launder and is led to a ground storage tank of 450 m^3 by gravity.

Another storage tank of 450 m^3 handles purified electrolyte.

There are six electrolyte coolers with induced draft fans. The spent electrolyte after cooling is mixed with neutral electrolyte and about 700 m^3 of feed is maintained in the main launder to feed 34-36 cascades (340-360 cells).

Anode cleaning is by tapping and chiseling out the manganese deposition. High pressure water is used to clean all the headers when once the cells are ready for electrolysis.

14.1.2 Measurements

Voltage drop and temperature measurements were carried out in detail. Voltage drops in rectifier room, busbars joints, below cell house (on each cascade joints), cells, anode-cathode to bus drops etc. have been taken using various meters. Necessary data has also been taken from daily production log sheets, logbooks, past data and office manuals made available.

14.2 ANALYSIS AND FINDINGS

14.2.1 Specific Energy Consumption

The average specific energy consumption per ton of zinc produced is 3492 kWh ; (Low : 3437 ; High 3568 kWh). The annual production is 28,387 tonnes.

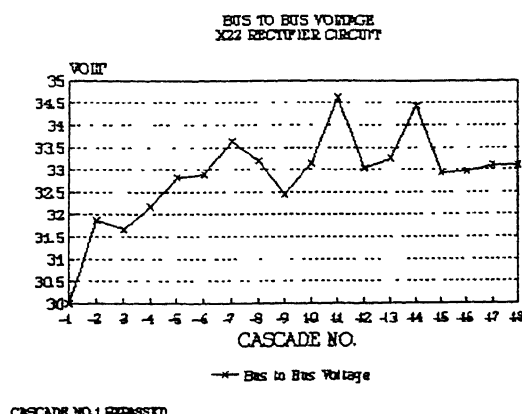
The monthly variation of production and total power consumption of electrolysis plant are depicted in graph for the year 1994-95. Details are given in Appendix 14.1/1

For analysis, production from sample cells were taken which were termed good or poor. Appendix 14.1/1 gives the production and consumption figures (cellswise for 3 cells). It may be observed that the specific energy consumption figures are low for cell no.5, average in cell no.17 and high for cell No.20.

SPECIFIC POWER CONSUMPTION

Days	Cell No.5 kWh/MT	Cell No.17 kWh/MT	Cell No.20 kWh/MT
28.7.95	3190	3619	3808
29.7.95	3232	3488	4083
30.7.95	2812	2996	3657
31.7.95	3145	3722	3934
01.8.95	3538	3862	3527
02.8.95	3165	3307	4263
03.8.95	3477	3589	3924

14.2.2 Cell Voltages and Bus to Bus Voltages

Individual cell voltages of 17 cascades were measured twice for X-22 circuit and once for X-12 circuit. Details of measurements are given in Appendix - 14.1/2. The exhibit beside shows variation of bus to bus voltage measured on X22 circuit.

The cell voltage measurements of X-22 circuit were taken during :

a. When anodes and cathodes wee frequently lifted and lowered after stripping and cleaning.

b. When the cells were ready for electrolysis.

It is observed that there are large variations in voltages during a. above and the duration of the process is almost 8 hours. This is unavoidable since all the lifting and cleaning process is manual, taking time. There is a difference of 11.45 V between summation of individual cell voltage and summation of cascade bus to bus voltage. This is attributed to the voltage drops of various busbar joints below and above cell cascades ; However, the cell voltage were observed to be varying from 3.1 to 3.3 volts during stable condition of cells ie, during electrolysis. Measurements are detailed in Appendix 14.1/2.

Observations made with X-12 circuit are given in Appendix 14.1/3. The exhibit beside shows variation of bus to bus voltage measured on X-12 circuit.

RANGE OF CASCADE VOLTAGES AS MEASURED (When Production was Steady)

Range of cascade voltages	No. of cascades			
	X-22 Circuit		X-12 circuit	
< 32 Volts	2	5	1	4
> 32 & < 34 Volts	13	12	16	13
> 34 Volts	2	-	-	-
Total circuit voltage	561.29	549.74	557.77	548.37

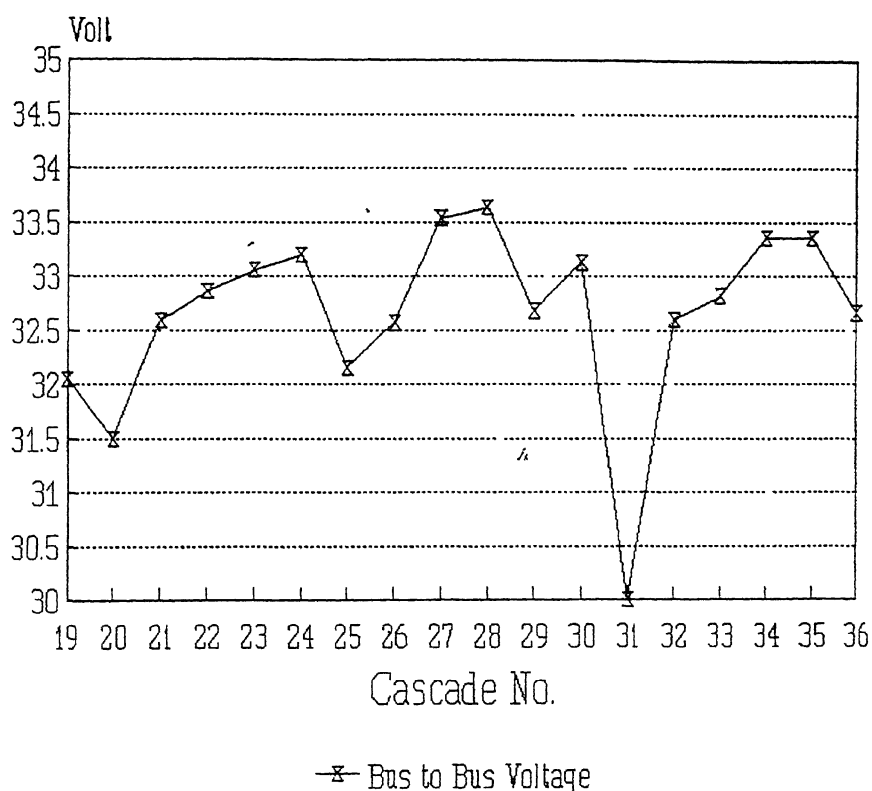
Millivolt drops across copper bus joints in rectifier room were measured. The description of joints are given in Sketch 1 A & 1B. The mV drops across joints in **rectifier room** may be measured regularly and corrective action may be initiated to minimise these drops.

CASCADE NO.31 BYPASSED

Figure 14

Detailed measurements are given in Appendix 14.1/4.

14.2.3 Analysis of Anodic and Cathodic mV drops

Millivolt drops across anodic and cathodic joints from copper bus were measured for few cells in cascade nos.5 and 17 (5A2, 5A3, 5B3, 17A4, 17B5) of X-12 circuits. Analysis is given in table below :

X-12 Circuit	Bus to Anode/Cathode mv drop measurements							
Cascade cell no.	Anode				Cathode			
	Min.	Max.	Avg.	Power loss kW	Min.	Max.	Avg.	Power loss kW
5 - A2	15.0	93.4	29.8	0.37	11.9	61.40	31.9	0.37
5 - A3	15.0	81.7	33.2	0.42	26.9	137.2	71	0.89
5 - B3	20.3	52.5	43.4	0.54	11.8	92.0	32.8	0.41
17 - A4	15.3	58.4	34.1	0.43	19.8	186.0	48.9	0.61
17 - B5	19.4	58.0	39.3	0.51	12.6	70.6	26.5	0.34

Load in kA = 12.5 to 13.0

The details of millivolt drops taken with reference to the 28 anodes and 27 cathodes for all the above referred cells are given in Appendix 14.1/5 (A, and B).

Similarly cells in cascade nos. 20 and 27 were sampled out and analysis of the same are give below :

X - 22 CIRCUIT- BUS TO ANODE - CATHODE mV DROP MEASUREMENTS

X-22 Circuit	Bus to Anode/Cathode mv drop measurements							
Cascade cell no.	Anode				Cathode			
	Min.	Max.	Avg.	Power loss kW	Min.	Max.	Avg.	Power loss kW
20 - B5	11.0	54.1	36.0	0.47	11.7	95.0	42.4	0.55
- A5	15.6	52.1	30.7	0.41	9.7	59.0	35.0	0.46
- A2	6.9	95	35.6	0.46	9.55	114	28	0.36
27 - A2*	18.0	113.1	45.61	0.59	55.29	101.4	70.24	0.91
- A4	8.94	73.68	37.65	0.49	55.1*	110.7	69.52	0.9
- B4	15.5	54.09	38.85	0.51	17.63	81.0	43.01	0.56
- B5	7.9	111.34	35.4	0.46	22.13	70.65	44.8	0.58

Load in kA = 13.0 kA

* The mV drops across anodic/cathodic contact joints to bus were observed to be high.

Details of anodic/cathodic drops are highlighted in Appendix 14.1/5 (C and D).

During the measurements and discussion, the anodes and cathodes having more than 70 mV drops were shown to the operator/supervisor, the contacts were cleaned and gap adjustment was rectified as per experience. The mV drops got reduced from very high values of 70-100 and above to 60 mV and below. In some cases the headers/bars were observed to be pitted, needing immediate replacement.

The copper tips used for contact are 99.9 % pure with a dimension of 40 x 15 x 10 mm, and this is brazed to the main header. This is being lifted thoroughly by the operator. The above anodic and cathodic power drops are quantified as given below : (Refer Appendix 14.1/6).

	Cascade No.	Power loss (kW)
X-12 Circuit	5	10
	17	9.45
X-22 Circuit	20	9
	27	12.5

Totally loss due to Anodic & Cathodic millivolt drops (In 17 cascades)

i. X-12 Circuit = 165 kW
ii. X-22 Circuit = 183 kW

It was observed during measurements that the gap between anodes & cathodes were not uniform. Generally 65 mm gap has to be maintained, but this was distorted in many places. When this was shown on the shop floor, the same were rectified and the mV drop values got reduced in some cases. Use of spacers for anode/cathode gaps is recommended.

Spacers made of material like ebonite/moulde can be inserted at the cell busbar side for a - 10 groups of anodes/cathodes so that the ga be set when once cells are ready for electro There is scope to minimise the same as gi recommendations.

14.2.4 D. Measurement of mV Drops Across Busbar Joi

Pure copper busbars are used above cells to c in series, the sections of (A & B) of each c containing 5 cells (Ref. Sketch 1 a and 1 cascade.

Similar joints are used below the cascac connect all the cascades in series. When cascade is taken for maintenance, a cross placed across the A/B sections of casca bolted.

An effort was made to measure all the joint drops above and below the cascades. Detai given in Appendix 14.1/7. Analysis measurements are summarised below.

X-22 CIRCUIT

Sl No	In voltage limits (mV)	Cascade Nos.	
		Cell top bus	Cell bottom
1.	< 10 mV	4A, 5A, 8A, 9A, 9B, 10A, 16 A/B, 17A, 18 A/B,	Nil
2.	> 10 mV < 30 mV	1A, 2 A/B, 3 A/B, 4B, 5B, 6B, 7 A/B, 8 B, 10B, 11A, 12 A/B, 13 A/B, 14 A/B, 15 A/b, 17 B	3 A/B, 4B, 6B 9B, 10 A/B, 1 14B, 15 A/B, 17 B,
3.	> 30 mV < 70 mV	1 B, 11 B	1 A/B, 4 A, 5 7B, 8 A/B, 9A A/B, 13 A/B, 16A, 17 A, 18
4.	> 70 mV	Nil	2 A/B, 12 A

X-12 CIRCUIT

Sl No	mV Limits	Cascade Nos.	
		Cell top bus	Cell bottom bus
1.	< 10 mV	19 A/b, 20 A/B, 22 A/B, 23-26 A/B, 27A, 30 A/B, 32 A/B, 33A, 35 B	34 B
2.	> 10 mV < 30 mV	21 A/B, 27 B, 28 A/B, 29 A/B, 31 B, 33 B, 34 A/B, 35 A, 36 A/B	34A, 35 A/b, 36 A/B
3.	> 30 mV < 70 mV	Nil	19 A/B, 20 A/B, 21 B, 22 A/B, 23 A/B, 24A, 25B, 26B, 27A, 28A/B, 29A, 30B, 31 A/B, 32A, 33 A/B,
4.	> 70 mV	Nil	21A, 24B, 25A, 26A, 27B, 29B, 30A, 32 B

Further analysis and summary of the data can be summarised & tabulated as below :

mV drop limits	No. of Bus Section Joints			
	X-22 Circuit		X-12 Circuit	
	Cell Top	Cell Bottom	Cell Top	Cell Bottom
< 10 mV	11	0	21	1
> 10 mV < 30 mV	22	14	14	5
> 30 mV < 70 mV	2	18*	0	22*
> 70 mV	0	3	0	8

* Indicates no. of high millivolt drop cascade sections

It may be observed that the millivolt drops of joints above cells for both D.C. circuits are fairly within limits. Almost all the 70 out of 72 bus joints have mV drops below 30 mV. Whereas, the mV drops below cell busbar joints are high. About 21 bus joints in X-22 circuits and 30 bus joints in X-12 circuits have voltage drops above 30 millivolt to as high as 110 mv.

The above observations indicate the need for regular monitoring of mV drop across joints especially on bus joints below the cells.

Generally overheating of bus sections and dust accumulations at joints were observed in most of the joints below the cells.

Details of calculations and power loss due to the above joint drops are calculated. (Given in Appendix 14.1/7) and summarised below :

Details	X-22 Circuit	X-12 Circuit
mV drop at joints above cells	593.42 mV	424.9 mV
mV drop at joints below cells	1415.60	1928 mV
Power loss kW	26.1	30.59

14.2.5 Power Loss Due to Resistance of Anodes and Cathodes

Anodes are made of pure lead, with 40.6 Sqcm area and cathodes are having a cross sectional area of 30.5 Sqcm. Details of dimensions and resistances are given in Appendix 14.1/8. Calculations of losses due to resistance of the electrodes are given below :

Details	Anode	Cathode
kW loss in electrode /cell	9.8	2

14.2.6 Power Loss Due to Electrolyte Resistance in the Cascade

An effort has been made to evaluate the total power loss due to electrolyte resistance in a cell. The total resistance offered for the flow or 13 kA current in the cells is the combined sum of resistances of anode and cathode and the electrolyte itself.

The current flowing from each anode face to cathode is 232 Amps and the power loss due to electrolyte is calculated to be 10.9 kW/cell ie., 109 kW/cascade.

Details of calculations are brought about in Appendix 14.1/9.

Summarising the calculated losses in sections 14.1.3 C, D, E & F, the measured losses for a typical cascade is given below :

Sl. No.	Loss Area (Hourly)	kW
1.	Bus-Bus contact power loss	26.1
2.	Anodic/Cathodic contact drops	10.0
3.	Resistance due anode/cathode electrode	11.0
4.	Loss in electrolyte	109.0
	Total	156.1

Details are given in Appendix 14.1/9.

It may be observed that Sl.No. 1 and 4 contribute to about 16% and 70% losses respectively. Hence it is advisable to :

a. periodically measure the contact mV drops of all bus joints and take corrective action of cleaning and retightening. This also reduces the temperature of bus section.

b. The temperature of the electrolyte is observed to be varying from 42 to 47 °C. This is a matter for improvement, since at lower temperatures, the resistance of the electrolyte drops & hence lower power drop. Details are given in Chapter 14.2.7.

14.2.7 Temperature of Electrolyte

The details of measurements, composition of feed electrolyte and spent electrolyte are given in Appendix 14.1/10. The average inlet and outlet temperatures of selected cells are given below :

Cell No.	Temperature °C		
	Inlet	Outlet	ΔT
5 A	37.33	44.46	7.13
B	37.67	42.10	4.43
17 A	37.60	47.57	9.87
B	37.80	44.40	6.6
20 A	37.06	47.6	10.54
B	37.20	47.5	8.33

The temperature rise of electrolyte is observed to be upto 47.6 °C and inlet temperatures are around 37.8 °C. This is on the higher side inspite of operation of five spent electrolyte coolers. It is advisable to cool the electrolyte to the maximum so that the feed temperature and spent electrolyte temperatures are lower.

The observations made on the spent electrolyte coolers are given in Appendix 14.1/11. The design drop being 6 °C, it was observed that coolers are having a range of 5.4 °C to 7.0 °C, with the operation of fans. However, the reasons for increase in temperature of electrolyte are due to losses within cells.

PIEGRAPH SHOWING BREAK-UP OF ELECTROLYSIS POWER

ELECTROLYSIS PLANT
BREAK-UP OF POWER

14.3 RECOMMENDATIONS

The following recommendations are made based on discussions referred in section 14.0 to 14.2.9.

A. Specific Energy Consumption and Monitoring of Cell Voltages

i. Presently the specific energy consumption of electrolysis plant is made based on available AC metering systems. However precision metering systems on AC and DC systems give the actual values of hourly energy consumption for meaningful analysis. Microprocessor based metering should be installed.

ii. At present there are no cell voltage monitoring systems. A common instrumentation panel indicating all the cell voltages and bus-bus voltages may be installed for better supervision and control.

B. Anodic and Cathodic Millivolt Drops

The copper bus to anode and cathode voltage drops are observed to be on the higher side. This varies from 15 mV (min) to 187 mV (max.).

From the observed values, a tolerance mill volt drop of 30-40 mV can be a guiding factor for further control of voltage drops.

By regular monitoring, gap adjustment and supervision, it is possible to minimise these voltage drops by 10% and hence minimise power losses. Reference to Sec 14.2.3 gives trials conducted on cascade nos. 5,17,20 and 27.

Power loss in X-12 circuit/hr = 165 kW
Power loss in X-22 circuit/hr = 183 kW

By improving supervision and monitoring by operators, it is possible to minimise losses and achieve energy savings on a continuous basis.

Annual energy savings = 2,50,560 kWh
Annual cost of savings = Rs.9,52,100
Annual cost of implementation = Rs.4,00,000 (Appx.)
(For manpower)
Simple payback period = 5 Months

C. Cascade Bus to Bus Series Millivolt Measurements

The mV drop values of bus to bus joints should be brought to average values by regular cleaning, and measurement (monitoring). It is estimated that 30% reduction in total mV drop can be reduced in X-22 circuit and 50% mV drop may be reduced in X-12 circuit.

Savings in energy losses/yr = 1,29,074 kWh
Cost of energy savings/yr = Rs.4,90,480/-
Cost of implementation = Rs.2,00,000/-(approx)
(For labour)
Simple payback period = 5 Months

14.4 SUMMARY OF POTENTIAL SAVINGS

Sl. No	Proposal	Estimated Energy Savings		Cost savings Rs.	Cost of implemen -tation Rs.	Simple payback period (years)
		Thermal kL/yr	Electrical (kWh/yr)			
1	Monitoring of Anodic and Cathodic millivolt drops	-	250560	952100	4,00,000	0 4
2	Measurement of bus to bus millivolt drop and maintenance	-	129074	490480	2,00,000	0 4
Total		-	379634	1442580	6,00,000	0 4

14.2 ZINC MELTING FURNACE

14.2.1 FACILITY DESCRIPTION

Zinc sheets from the stripping platform weighed and charged into two low frequency induction furnaces each having a holding capacity of 25 MT. (One is of Russian Make and the other AJAX make). The melting capacity of each of the furnace is about 5.4 MT/hr of zinc. Molten zinc is manually cast into slabs (22 kgs) and transported to storage yard.

14.2.2 OBSERVATIONS, ANALYSIS AND FINDINGS

A. Efficiency of AJAX and Russian Furnaces

Sample observations carried out on furnace temperature, electrical input parameters, material charged etc have been tabulated in Appendix - 14.2/1. Towards establishment of furnace efficiency, theoretical power requirement for zinc melting has been computed. Heat required to melt 1 ton of zinc works out to 76.0 kWh/MT of zinc. With the existing variation in power consumption, efficiency of melting works out to 54% to 63 %. Calculation details are given in Appendix 14.2/2.

B. Insulation Aspects

Towards assessment of radiation losses, measured variation of skin temperatures of both the furnaces are as given below :

Furnace Ref.	Measured surface temp. °C	Total heat loss kcal/hr
AJAX Furnace	51 to 72	6992.25
Russian Furnace	58 to 82	9181.65

Observed surface temperatures indicate adequate insulation provision and radiation losses within limits. Calculation details are given in Appendix - 14.2/3.

C. Zinc Pouring Temperatures

It is observed that zinc pouring temperature is maintained at around 465 °C to 470 °C. However maintaining a temperature of 450 °C would still have a margin of higher temperature of 15 °C to 20 °C which is undesirable.

The estimated power consumption for every 10 °C rise in melt temperature per ton of zinc is about 1.35 temperature of 450 °C should be adhered to.

D. Provision of door

Existing door of Russian furnace should be made operational as to close it during non-casting hours. This besides being a good operational measure would help in reducing radiation losses.

15.0 LEAD PLANT

15.0.1 FACILITY DESCRIPTION

Lead plant was commissioned in 1978 with an installed capacity of 10000 MT per year. The capacity of the plant was increased to 22000 MT by installing a new blast furnace in the year 1983.

Process of lead smelting involves sintering of lead concentrate in sinter machine followed by reduction of coarse sinters in blast furnace to produce bullion lead and refining lead in kettle by pyro refining. The process chart is given in Appendix - 15/1.

Lead smelter comprises of following major energy consuming equipments :

1. Sinter Machine
2. Blast Furnace
3. Slag Settler
4. Lead Refinery Kettles
5. Rotary Furnace

Production and energy consumption in the above equipments during 1994-95 are tabulated below :

Equipment	Production MT	LDO kL	FO kL	Coke MT
Sinter machine	37880	308	-	-
Blast furnace incldg. slag settler	12043	-	20	5075
Lead refinery	10143	835	520	-
Rotary furnace	838	25	256	-

15.1.0 SINTER MACHINE

15.1.1 FACILITY DESCRIPTION

Sinter plant is installed with a 20 m^2 lurgi design updraft sinter machine of capacity 30 MT/hour. The functional utility of sinter machine is to produce self fluxing sinters having high porosity and hardness for smooth reduction of lead oxide in blast furnace. The above properties can be achieved by converting sulphate and sulphites of lead into lead oxide and sulphur dioxide.

The input materials to the sinter machine are return sinters of size below 25 mm, granulated slag from slag settler, coke breeze, lime stone, iron ore and lead concentrate/scrubber dust.

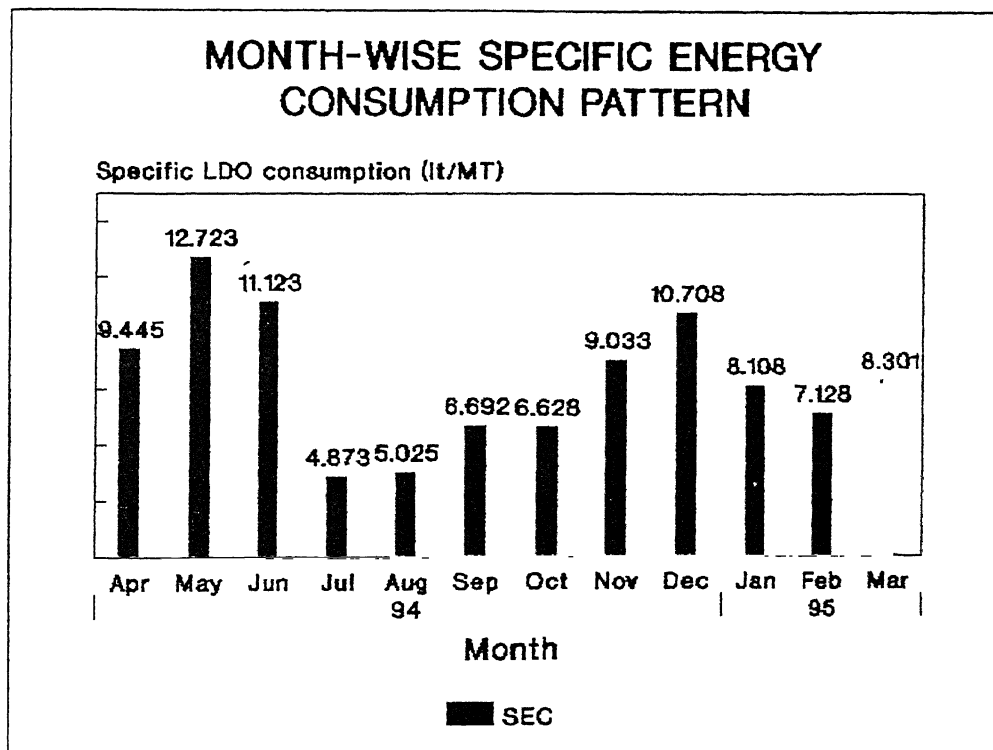
The output material consists of fine sinter of about 65% (of particle size below 25mm) and 35% of coarse sinter. The fine sinter is sent back to the machine while coarse sinter is transported to blast furnace for reduction.

Energy Consumption

LDO is used to ignite the sinter bed upto 35mm to initiate the exothermic reaction of the material. Little quantities of coke breeze (0.4 MT/hr) is added to sinter machine to convert scrubber dust into coarse sinter.

Monthly Energy Consumption and Production values are given in Appendix - 15.1/1. The minimum, maximum and average specific energy consumption values of LDO are tabulated below :

Particulars	Unit	Quantity
Avg.LDO consumption	kL/month	25.667
Avg production of coarse sinter	MT/month	3156.67
Sp.energy consn of LDO	L/MT	
Minimum		4.873
Maximum		12.723
Average		8.315

The following bar chart gives the monthwise specific energy consumption during 1994-95:

It can be seen that the specific energy consumption varied in the range of 4.873 - 12.723 L/MT. The factors contributing to wide variation are feed material composition, input moisture content, extent of exothermic reactions, etc.

15.1.2 OBSERVATIONS, ANALYSIS AND FINDINGS

A. Operational Features

The various material inputs to the sinter machine are feeder materials, air through combustion air blower and fresh air blower, water through drum mixers and furnace oil through burner. Total input material is estimated at 41.72 MT/h. Quantity of each material is estimated and tabulated in Appendix - 15.1/2. The following table gives the summary of inputs :

Sl No.	Material	Quantity MT/hr
1	Feeder material	26.55
2	Air through blowers	14.12
3	Water through drum mixers	1.00
4	Fuel through burners	0.05
	Total	41.72

All feeders are provided with speed control system by which the input rates of all materials can be controlled according to required composition of output sinter. An attempt was made to understand the operations of sinter machine by systematic observations of various parameters such as feed rate, air flow rates of fresh air, recirculation air and combustion air, gas temperature, wind box pressures, percentage SO_2 in exhaust gases, etc. The observed parameters monitored and recorded during the study are given in Appendix - 15.1/3.

B. Energy Balance

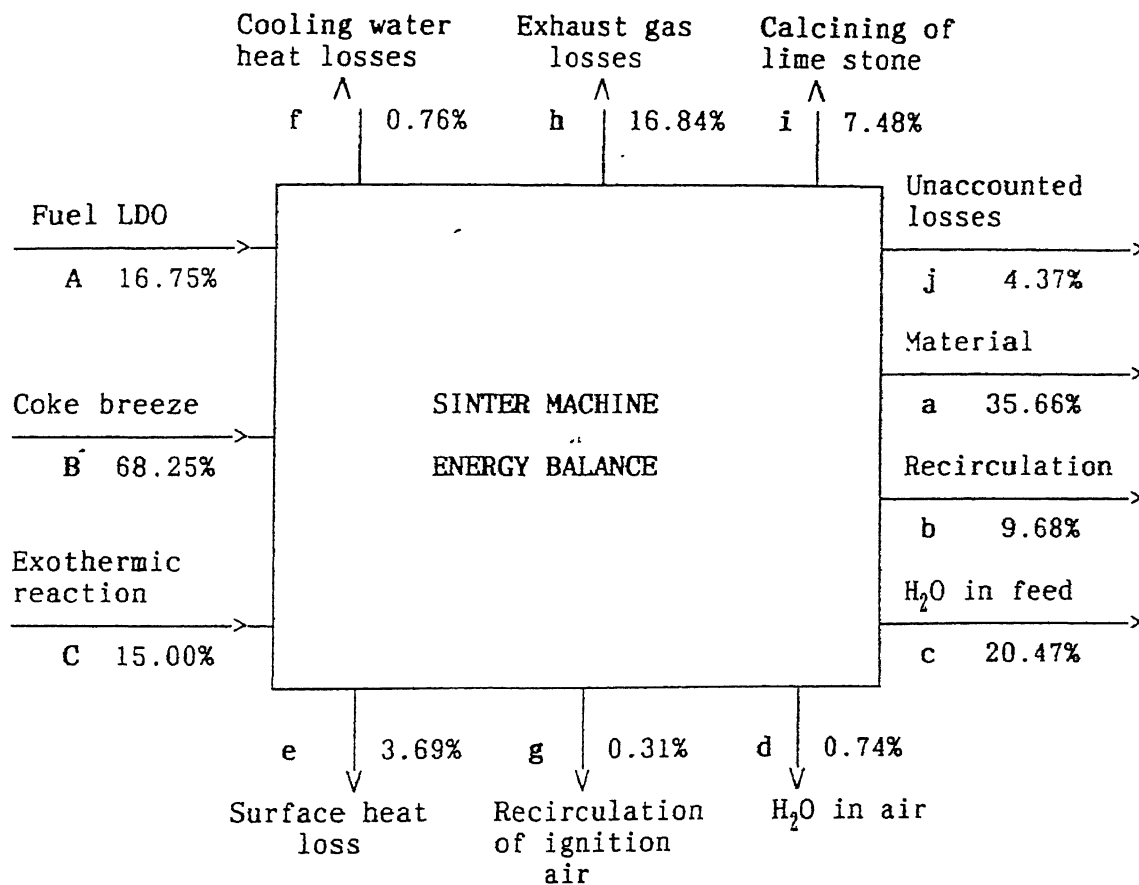
Energy balance of sinter machine has been carried out by quantifying various heat inputs and heat outputs as given below :

i. Heat Inputs

A. Heat given through fuel
B. Heat given through coke breeze
C. Heat of exothermic reaction

ii. Heat Outputs

a. Heat given to the material
b. Heat loss due to recirculation of air
c. Heat loss due to H_2O in feed material
d. Heat loss due to H_2O in air
e. Heat loss due to surface heat losses
f. Heat loss due to vertical and horizontal jacket cooling by water
g. Heat loss due to recirculation of ignition air
h. Heat given to exhaust gases
i. Calcination of lime stone
j. Unaccounted losses

The detailed quantification of all heat inputs and outputs are given in Appendix - 15.1/4. The following diagram depicts the percentage contribution of various heat inputs and outputs :

From the above, it can be seen that the sinter machine is operating at 35.66% efficiency. The other major losses are heat loss due to water in feed material, surface heat losses, recirculation losses, and exhaust gas losses. The possibilities of reducing heat losses and recovering waste heat were explored to increase efficiency of the sinter machine. These are discussed in the following sections :

C. Heat loss due to Water present in the Feed Material

Water, about 500 kg/h per stage is added to the charge material in two stages viz., during charge preparation and before feeding the material to sinter machine. The purpose of water addition in first stage is to avoid dust evolution during transit of material and addition in second stage helps in forming granules. The final water content in the feed material should be in the range of 5-6%. Heat loss due to moisture content accounts for 20.47% of total heat input ie., 660000 kcal/h. This loss cannot be avoided or reduced as the presence of moisture is essential.

D. Surface Heat Losses

Surface heat losses of sinter machine were quantified by sectionalising the surface area into different zones and measuring surface temperatures and areas. The observed temperatures were found to be in acceptable range. The quantified total heat losses and heat losses per m^2 are summarised in the following table:

Particulars/ section	Total heat loss kcal/h	$kcal/h.m^2$
Sinter machine	98327	1252.58
Material outlet chamber	2068	1175.45
Total	119015	1238.46

The total surface heat losses account for 3.69% total heat input. The reduction in heat losses will not result in energy savings due to requirement of material cooling in the machine.

E. Exhaust Gas Losses

Total quantity exhaust gases vary in the range 8000-12000 m^3/hr, which contains SO_2 of 1-2%. These exhaust gases are sucked by a blower and passed through gas cleaning plant and finally sent to H_2SO_4 plant. The temperature of exhaust gases varies in the range of 170-250°C. The heat lost in exhaust gases contribute to 16.84% of total heat input (ie., 542810 kcal/hr).

Low outlet temperatures of exhaust gases, presence of sulphur dioxide and high dust content (30 g/Nm^3) of gas restricts the heat recovery from exhaust gases. High SO_2 content and low gas temperature leads to condensation of H_2SO_4 and cause corrosion of metal. Dust in exhaust gases results in choking and deposition on the surfaces. The above factors do not allow recovery of the heat from exhaust gases, even though the amount of heat carried away by exhaust gases is large.

F. Heat loss due to Recirculation of Air

About 146.2 Nm^3/min of air in the sinter machine is being recirculated through long duct to cool the material in the furnace and enrich gases for SO_2. The heat loss due to the recirculation is estimated at 312051 kcal/hr amounting 9.68% of total heat input.

This heat rejected to the atmosphere cannot be recovered due to practical problem of waste heat recovery such as :

- dust content in the recirculating air
- high SO_2 content in air, which leads to condensation whenever the temeprature falls below sulphur dew point. More over duct cannot be insulated since cold recirculation air is required to cool the feed material.

G. Use of FO in Burners in Place of LDO

The ignition chamber of sinter machine is provided with two burners on either side. One of these burners uses LDO and another furnace oil. During normal operation of sinter machine only one burner will be firing and another kept as standby. Based on the past data (for the year 1994-95) provided to the audit team, it was observed that LDO was used though the machine is provided with FO burner.

Comparative analysis has been carried out to use FO continuously and keeping LDO as standby. The summary of analysis is tabulated below :

Particulars	Unit	LDO	FO
Specific cost of energy	kcal/Re	1255.81	1812
Hourly consumption of fuel	L/h	58.83	55.75
Hourly cost of heating	Rs./h	430	307*
Cost savings	Rs.lakh/yr	-	8.656
Equivalent LDO Savings	kL/year	-	121.14

*Includes preheating cost

It can be seen that the existing hourly cost of LDO firing worked out to Rs.430/- while in the case of furnace oil, it is Rs.307/-. Appreciable cost savings (ie., Rs.8.856 lakh) can be achieved by using FO in sinter machine without any investment. Detailed calculations are given in Appendix - 15.1/6.

H. Efficiency of Fans

Sinter machine is installed with two fans viz., fresh air fan and recirculation fan. Fresh air fan supplies ambient air to the machine to speed up the oxidisation of feeder material. Recirculation fan is used to recirculate the air in the sinter machine to enrich lean gases for SO_2 and to cool the gases (inturn to cool the material) by rejecting heat to atmosphere during circulation.

Parameters such as pressure, power consumption, air flow were monitored to evaluate the output and efficiency of the fan. The evaluated efficiency and output are tabulated below :

Fan	Output, m^3/min	Percentage Output	Efficiency, %
Fresh air	172.0	86.0%	58.11
Recirculating	146.2	73.1%	-

It can be observed that the fresh air and recirculation fans are operating at 86% and 73.1% of their rated capacities respectively. The combined efficiency of fresh air fan and motor is estimated at 58.11%. The detailed calculations are given in Appendix - 15.1/7.

J. Metering of Oil Consumption

One LDO service tank of 12 kL capacity is being used to meet day to day needs of sinter machine. The consumption of LDO is being estimated based on rated consumption of burner and number of operating hours. The present method of metering can give only approximate oil consumption, if the burner operates continuously and at maximum valve opening.

The appropriate oil metering is essential to monitor the exact oil consumption and variation in oil flow. The variation in flow rate may be due to the process requirements or poor burner performance. By metering the consumption, the burner performance can be evaluated and thereby remedial action can be taken to reduce the oil consumption.

The two methods of metering are dipstick method, in which the level difference of oil in the tank during every 24 hours is monitored, and the other method is by installing a flow meter near the burner.

15.1.3 RECOMMENDATIONS

A. Use of Furnace Oil in Place of LDO in Sinter Machine

There exists a good potential in cost savings by using furnace oil continuously and keeping LDO as standby. By implementing this measure, the cost of hourly oil consumption can be reduced to Rs.307/- from existing Rs.430/-. Refer Section 15.1.2 (G) for details.

Estimated savings

Savings in LDO	= 121.14 kL/year
Cost savings	= Rs.8.856 lakhs/year
Investment required	= Nil
Simple payback period	= Immediate

B. Metering of Oil Consumption

Metering of oil consumption should be practised and monitored regularly. Consumption metering can be done by using either dipstick method or by using flowmeter.

15.1.4 SUMMARY OF POTENTIAL SAVINGS

Recommendation	LDO Savings kL/yr	Cost savings Rs.lakhs	Investment required Rs.lakhs	Payback period (yrs)
Use of furnace oil	121.14	8.856	Nil	Immediate
Total	121.14	8.856	Nil	Immediate

15.2.0 BLAST FURNACE

15.2.1 FACILITY DESCRIPTION

The blast furnace is of 4.8 m height with a designed hearth area of 4.68 m^2 and equipped with 30 tuyers. The furnace is designed to produce 86 MT/day of hard lead.

The blast furnace is used for reducing PbO into Pb and CO_2 and lead melting. The molten lead and slag are tapped through separate spouts. There are three bins of 30 m^3 capacity for storing coarse sinter. The other two bins are for slag and hard coke. The material is drawn from these bins and fed to blast furnace in the following ratio.

Coarse sinter	=	4 MT
Coke hard	=	700 kg
Slag	=	150 kg
PbOH	=	40 kg
No.of charges/shift	=	16

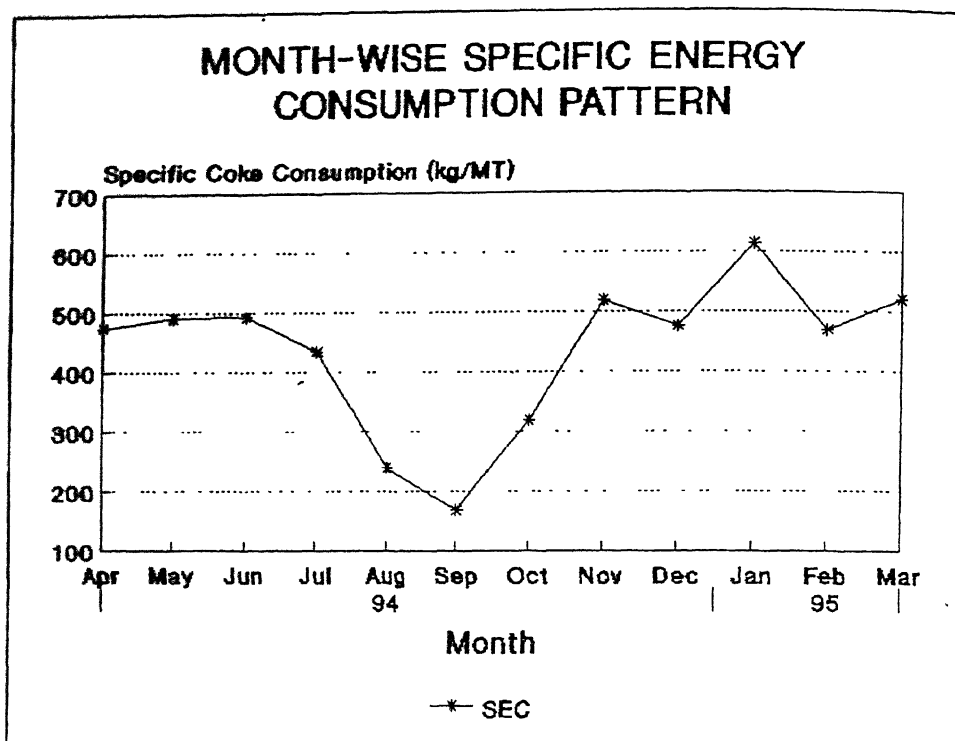
The outlet material consists 35% of hard lead and 65% slag of total sinter input.

Energy Consumption

Hard coke is used as reducing media and energy source to melt lead. During 1994-95, the total coke consumption was 5075 MT. Month-wise coke consumption along with specific energy consumption is given in Appendix - 15.2/1.

The following graph indicates the variation in the specific energy consumption (kg/MT of lead).

The average coke consumption (kg) per metric ton of lead tapped is estimated at 434.672, while the minimum and the maximum values of specific coke consumption are 168.440 and 613.158 respectively.

The wide variation in specific coke consumption is due to variation in inlet sinter composition and extent of endothermic and exothermic reactions in the blast furnace.

15.2.2 OBSERVATIONS, ANALYSIS & FINDINGS

A. Operational Features

During the audit, every attempt was made to understand the operational features of blast furnace. Various parameters such as gas temperature, air flow rate, oxygen flow rate, cooling water outlet temperatures, lead temperature, slag temperature, CO_2 percentage, charge composition were monitored and recorded. These parameters along with design operating parameters are given in Appendix - 15.2/2.

The summary of observed parameters are given in the following table :

Parameters	Average
Flue gas temperature °C	445.0
Air flow rate Nm^3/hr	5933.0
Air pressure mm wg	1316.6
Oxygen flow rate Nm^3/hr	50.00

Parameters	Average
Mantel Jacket water outlet temp.°C	48.02
Main Jacket water outlet temp.°C	51.60
Chute Jacket water outlet temp.°C	45.0
Channel Jacket water outlet temp.°C	45.3
Lead temperature °C	1073.3
Slag temperature °C	1358.3
CO_2 in flue gas %	17.5

It was observed that the operating parameters are within reach of design range except temperatures of outlet lead and slag. The design outlet temperatures of lead and slag are 800-1000°C and 1100-1200°C respectively.

B. Energy Balance of Blast Furnace

The energy balance of blast furnace has been attempted at by quantifying various heat inputs and heat outputs such as:

Heat Inputs

a. Heat given through hard coke

Heat Outputs

a. Heat given to lead
b. Heat given to PbO to form lead
c. Heat given to slag
d. Heat given to cooling water
e. Flue gas losses
f. Surface heat losses
g. Heat loss due to unburnts (CO_2) in exhaust gases
h. Heat given for complex reactions and unaccounted losses

Observation and measurements of various parameters attempted at for estimation of the above losses are mentioned below :

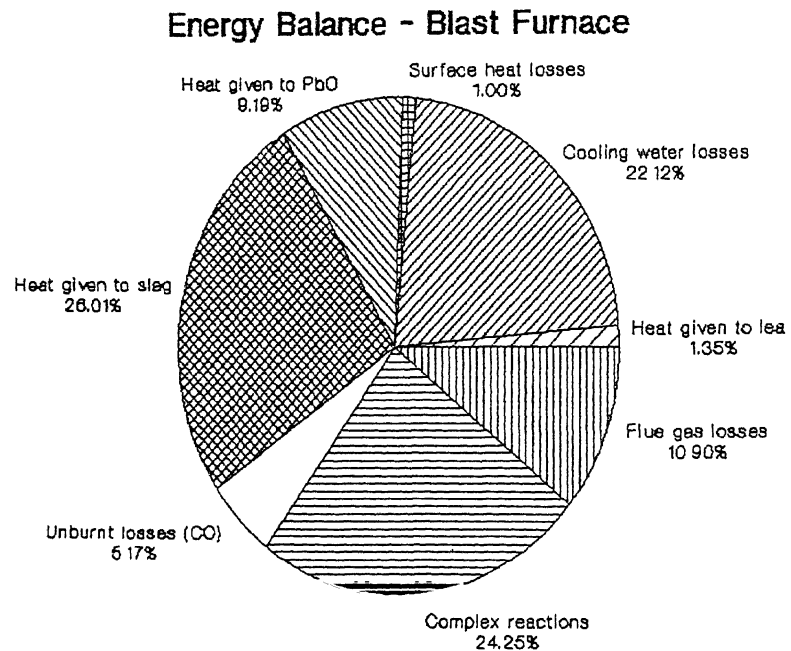
i. Temperatures of cooling water outlets and inlet, lead, slag and ambient air

ii. Weights of input materials and output lead

iii. Surface temperatures of blast furnace

iv. Flow rate of blast air

Detailed quantification of heat components are given in Appendix - 15.2/3.

The following pie diagram indicates percentage contribution :

The useful heat outputs are heat given to lead, PbO for reduction, slag and complex reactions, which together accounts for 60.80% of total heat input ie., 4682074 kcal/h.

The major heat loss components are cooling water losses and flue gas losses. Possibilities were studied either to reduce the losses or recover the losses. These are discussed in the following sections.

C. Cooling Water Losses

Cooling water is used in jackets of main frame, mantle, chutes, channel and slag input. The number of cooling water jackets in each application are:

Application	No.of jackets
Main jacket	18
Mantle jacket	22
Chute jacket	2
Channel jacket	2
Slag spout jacket	1

Trials were conducted to estimate cooling water flow rate in each jacket by collecting the water in a measured bucket. The measured values of water flow rates are tabulated in Appendix - 15.2/4. Total cooling water flow rate in blast furnace is estimated at 153.34 m^3/h. Heat loss due to cooling water circulation is estimated based on water flow rate and rise in water temperature (Refer Appendix -15.2/3)

The summary of cooling water flow and corresponding heat losses are:

Jacket	Cooling water flow kg/hr	Heat loss kcal/hr
Mantle	27676	277313
Main	81234	1105594
Chute	27030	197319
Channel	15900	105630
Slag	1500	18000
Total	153340	1703856

Total cooling water losses account for 22.12% of total heat input. These losses are inevitable to avoid fusing of tuyer and respected Jones.

D. Flue Gas Losses

The measured flue gas temperatures varied in the range of 440-450°C and the corresponding flue gas losses were estimated at 838930 kcal/hr (ie., 10.9% of total heat input). High temperature flue gas and heat content of exit flue gases indicates scope for heat recovery by preheating the combustion air.

Possibility of using recuperator for the purpose was studied. Since the exhaust gas contains high level dust which is sticky in nature, restricts the heat recovery.

It was also understood that the exhaust gas lines are subjected to frequent choking due to the dust, and plant personnel clear the dust as and when choking occurs.

E. Efficiency of Roots Blower

Roots blower supplies combustion air to blast furnace through tuyer. Performance of roots blower has been studied. For the purpose of analysis, parameters such as power consumption, outlet air pressure, air flow rate were monitored. The combined efficiency of motor and blowers was estimated at 42%. The detailed calculation are given in Appendix - 15.2/5.

The normal operating efficiency of roots blower (motor and blower combined efficiency) at specified rated conditions vary in the range of 50-60%. The factors which are contributing for low efficiency may be operating condition of blower, outlet damper control for air flow , etc.

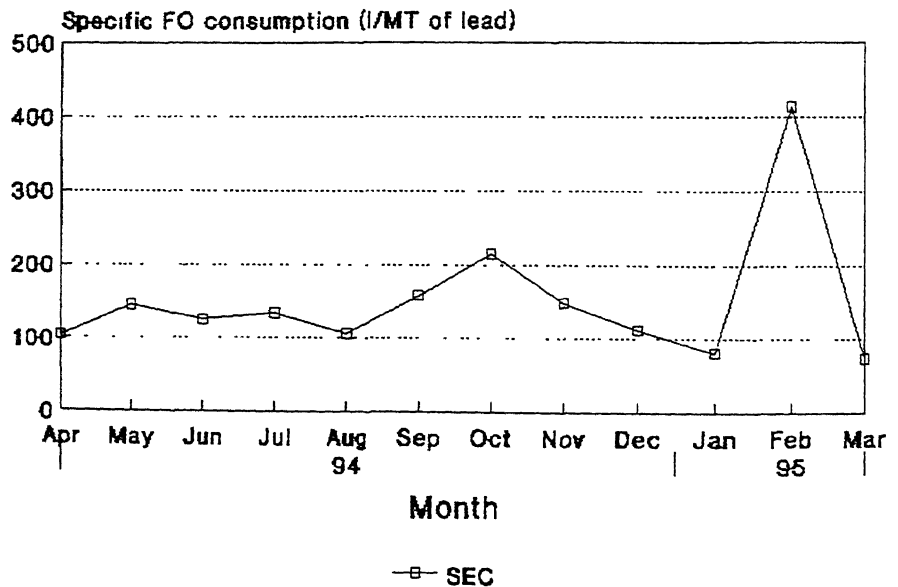
15.3.0 SLAG SETTLER

15.3.1 FACILITY DESCRIPTION

The slag having lead of 1-2% from blast furnace is tapped intermittently and conveyed into furnace oil heated slag settler to separate the lead and slag. The molten lead is tapped from the bottom of the tank and casted in swivelling launder. The slag overflow from the tank is granulated by water spraying.

The settling tank is installed with two burners and can use either furnace oil or LDO. During normal conditions, furnace oil is used in the burners.

Energy Consumption

During the year 1994-95, the slag settler has consumed 283 kL of furnace oil. The specific oil consumption is evaluated based on blast furnace production. Since the slag settler is being connected to blast furnace. The following graph gives the variation in specific energy consumption.

The Specific energy consumption varied in the range 14.093 - 33.019 l of FO/MT. The average specific FO consumption per MT of lead production during 1994-95 is 25.285. The monthly consumption values along with production are given in Appendix - 15.3/1.

15.3.2 OBSERVATIONS, ANALYSIS AND FINDINGS

A. Operational Features

Once in every 20 minutes, the slag spout of blast furnace is opened and the slag is transferred to settler through a refractory channel. This slag is heated in the slag settler using furnace oil to melt the lead and slag. Molten lead will settle down at the bottom due to high density and slag floats over the lead. Settled lead (about 200 kg/shift of lead) is tapped from the bottom spout of slag settler. The overflown slag is granulated by water spraying. The various parameters such as slag inlet temperature, cooling water inlet and outlet temperatures, flue gas temperatures, oil pressures, combustion air flow rates were monitored and recorded during the audit study. These parameters are placed in Appendix - 15.3/2.

B. Energy Balance

Slag Settler was studied for energy balance. Various losses such as flue gas losses, surface heat losses, loss due to openings and cooling water losses were estimated. The following table gives the summary of energy balance.

Particulars	kcal/hr	Percentage
Heat Input	499290	100.00
Heat Output		
Flue gas losses	184844	37.02
Cooling water losses	143100	28.66
Losses due to opening	24756	4.96
Surface heat losses	23327	4.67
Efficiency & unaccounted losses	123263	24.69

The settling tank was operating at 24.69% efficiency. The detailed calculations are given in Appendix - 15.3/3.

15.4.0 LEAD REFINERY PLANT

15.4.1 FACILITY DESCRIPTION

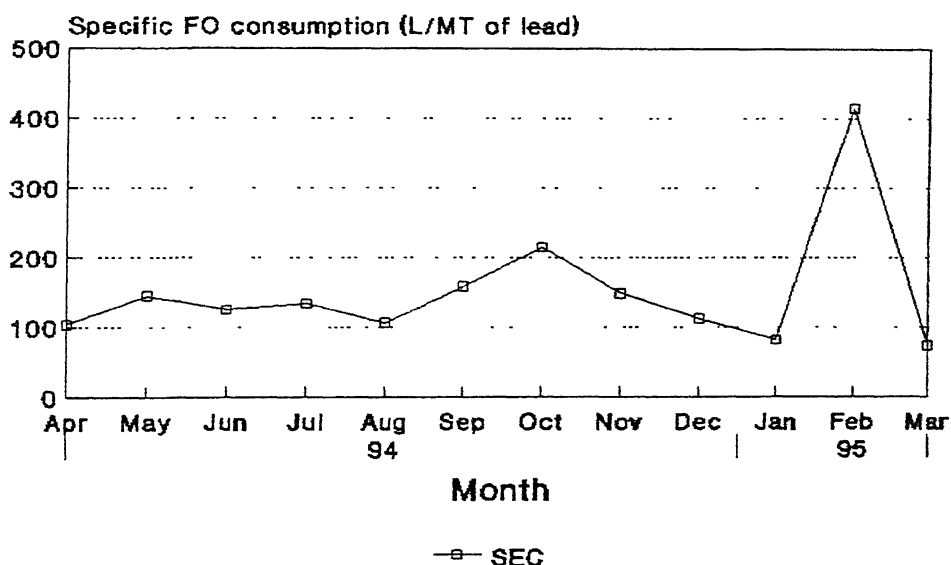
The lead refinery plant is equipped with eight kettle furnaces of 60 MT capacity for refining the hard lead of blast furnace. The lead is refined in a series of operations based on MIM technology which facilitates desilvering preceeding deantimonising. Brief process description of lead refinery along with desired process parameters are given in Appendix - 15.4/1.

The summary of the operations in lead refining are tabulated in the following table :

Kettle No.	Activity	Salient Features
1 & 2	Ordinary drossing & decopperisation	Melting time upto = 500 hours Fuel consumption (melting) = 6 7 lts/t Decopperisation time = 24 hours Fuel consumption during decopperisation = 5/t
4	Dearsonating	Caustic soda addition = 0.5 kg/t Fuel required = 3.1 l/t Temperature range = 470-490°C Cycle time = 6 hours
5	Desilverisation - I stage Desilverisation - II stage	Zinc addition = 11-12 5 kg/t Final silver content = 5-10 g/t Cycle time I stage = 7-9 hours II stage = 15-16 hours Fuel consumption I stage = 5 0 l/t II stage = 3.4 l/t Temperature = 460°C
6	Dezincing	Cycle time = 13 hours Zinc after dezincing = 0 05% Zinc recovery = 90% Fuel consumption = 6 4 l/t Temperature = 590°C Min vacuum = 0 5 um
7	Softening	Caustic soda addition = 8 4 kg/t Sodium Nitrate = 2 8 kg/t Fuel consumption = 9 0 l/t Temperature = 480-490°C
8	Casting	Cycle time = 8 hours Temperature = 450°C (min)

ENERGY CONSUMPTION

Furnace oil and LDO are the major fuels used in refining kettles. During 1994-95, the LDO and FO consumption was 791.053 kL and 1311.329 kL respectively. Since refinery uses dual fuels, specific energy consumption values were evaluated by converting LDO consumption into equivalent furnace oil consumption by considering calorific value and specific gravity. The following graph gives the specific furnace oil consumption values during 1994-95.

MONTH-WISE SPECIFIC ENERGY CONSUMPTION PATTERN

he average furnace oil consumption during 1994-95 per MT of lead produced was 151.387 L. The high specific energy consumption during February 1995 (ie., 414.149 L/MT) was due to very low production during that month. Details of month-wise energy consumption and production are given in Appendix-15.4/2.

15.4.2 OBSERVATIONS, ANALYSIS AND FINDINGS

A. Combustion Efficiency

Various factors which affect the efficiency of the kettles and fuel consumption, such as combustion efficiency, flue gas losses, excess air, surface heat losses, heat losses through openings were studied and analysed for various energy conservation measures.

For the purpose of analysis, parameters such as temperatures of surfaces, ambient dry and wet bulb, flue gas, furnace oil and metal, CO_2 percentage in flue gas, furnace oil consumption, etc were monitored and recorded.

The efficiency of the kettle furnaces were evaluated by adopting indirect method, in which various losses are deducted from the total heat input to arrive at the useful heat and efficiency. The detailed calculations are given in the Appendix - 15.4/3. The outcome of the analysis is given below :

Kettle No.	Efficiency (% η)	Remarks (Efficiency)	Reasons for low efficiency
1	20.27	Low η	High excess air levels
2	20.23	Low η	High excess air levels
4	52.69	Moderate η	High flue gas temp. & high surface heat losses from top of the kettle
6	26.59	Low η	High excess air levels
7	37.76	Low η	High excess air levels and high flue gas temp.
8	18.58	Low η	High excess air level, high flue gas temp. and high surface heat losses from top of the kettle

All kettles were operating at low efficiency. The main factors contributing to low efficiency were high excess air levels, high flue gas temperature and high surface heat losses in few kettles, apart from heat losses occurring through door opening.

The various measures to reduce the losses and improve the efficiency are discussed in the following sections.

B. Excess Air Losses

The measured CO_2% in the flue gases of kettle furnaces indicated that the burners were operating at very high excess air levels of range 20-420%.

The stoichiometric air required for furnace oil combustion is 13.78 kg per kg of furnace oil and the maximum percentage of CO_2 in flue gas is 15.6. Furnace oil burners require 20-30% excess air level for efficient combustion and corresponding CO_2 percentage in flue gases is 12-13%.

Excess air above 25% will result in reduction of flue gas temperature which reduces the heat transfer rate from flue gases to kettle and unnecessary heating of excess air.

Heat losses due to excess air in kettle furnaces

Kettle	% CO_2	Excess air %	Heat loss due to excess air (per kg of fuel) kcal	% heat loss due to excess air
1	3	420	4526	44.37
2	3	420	4391	43.05
4	13	20	325	3.19
6	4	290	3578	35.08
7	6	160	2280	22.35
8	4	290	4116	40.35

It can be seen that excess air is resulting in heat losses as high as 44.37% of total heat input. All the furnace burners are installed with ratio controllers for air & fuel; and these are yet to be commissioned. These controllers are to be put into practice to reduce the excess air losses and maintain CO_2 in the range of 12-13%.

The proposed efficiency of kettles after controlling excess air is evaluated and placed in Appendix 15.4/4.

Proposed efficiency of kettle furnaces

Kettle	Excess air losses per kg of fuel kcal	% heat loss due to excess air	Present efficiency %	Proposed efficiency %	Improve-ment in eff.
1	268	2.67	20.27	70.44	50.17
2	260	2.55	20.23	68.91	48.68
4	325	3.19	52.69	52.69	00.00
6	306	3.00	26.59	65.15	38.56
7	354	3.43	37.76	60.46	22.70
8	352	3.46	18.58	62.92	44.34

C. Heat Recovery from Flue Gases

The high exit flue gas temperature and huge quantity of flue gases indicate potential for waste heat recovery by preheating the combustion air upto 250°C.

Feasibility was studied to recover the heat in exhaust gases after considering the excess air level of 20-30%.

Summary of Techno-economics of the measure is tabulated below :

Kettle	Recoverable heat, kcal/hr	Savings in FO, kL/year	Cost savings, Rs.lakh/year
1	98175	50.63	2.705
2	101178	52.21	2.790
4	98175	50.63	2.705
6	98175	50.63	2.705
7	98175	50.63	2.705
8	102148	52.63	2.812
5*	98175	50.63	2.705
Total	694201	358.00	19.127

During audit study, kettle No.5 was not in operation, hence lowest possible savings were considered.

The annual savings are estimated at 358 kL of furnace oil (ie., Rs.15.33 lakhs/year) with an investment of Rs.56.00 lakhs, it yields a simple payback of 3.6 years. Detailed techno-economics of the measure are given in Appendix - 15.4/5.

To implement this measure, existing blowers are to be replaced with high pressure ones. The cost savings are estimated after considering the increased power consumption in blowers. This measure can be tried on one kettle furnace and if found successful this can be carried out on remaining kettles.

After discussion with plant personnel and subsequent discussions held by the plant with suppliers, this measure was observed to be not practically feasible. Since the proposal is also techno-economically not viable, the proposal is dropped.

D. Surface Heat Losses

Heat losses from kettle furnace surfaces quantified by sectionalising them into different zones and measuring surface temperatures and areas. The quantum of surface heat losses depends upon the surface area, temperatures, ambient temperatures, orientation and emissivity of the surface.

Kettle-wise measured and quantified losses are tabulated in Appendix - 15.4/3. Summary of heat losses and average heat losses per unit area are given below :

Kettle	Total heat losses kcal/hr	% heat losses to heat input	Heat loss kcal/h.m^2	Remarks
1	62043	4.91	2197	Reasonable losses
2	86331	6.63	3057	Reasonable losses
4	181035	14.34	6410	High losses
6	95768	7.59	3391	Reasonable losses
7	115625	9.16	4094	High losses
8	89578	6.82	3172	Reasonable losses

The surface heat losses in most of the kettles are falling in the close range except in kettle No.4 and kettle No.7. High surface heat losses in kettle 4 and 7 are due to high material temperature in the kettle (during the measurement).

E. Heat Loss due to Door Openings

Kettle furnaces are provided with a single door to enable the operator to carry out maintenance work inside the furnace. These doors are kept open during the normal operation of burner to check the flame condition. The heat losses through door openings of furnace (ie., black body radiation) are estimated at 97363 kcal/h.

Provision of peep hole next to the burner for checking the flame and closing of door during burner operation avoid the heat losses through opening. Estimation of heat losses, savings after implementation of the measure are given in Appendix -15.4/6.

Kettle-wise heat losses and savings

Kettle	Heat loss kcal	Savings in FO kL/year	Cost savings Rs.lakh/year
1	58500	7.26	0.380
2	78000	9.74	0.520
4	78000	9.74	0.520
6	62400	7.79	0.416
7	62400	7.99	0.460
8	63180	7.89	0.421
Total	97363	50.21	2.681

The annual energy savings to the tune of Rs.2.680 lakh/year can be envisaged with a marginal investment.

15.4.3 RECOMMENDATIONS

A. Controlling Excess Air Levels in Burners

Excess air in burner of kettle furnaces can be brought down to 30% by commissioning the ratio controllers installed already in the system. The implementation of the measure will improve the efficiency substantially. Refer Section 15.4.2 (B) for details.

B. Avoiding Heat Loss through Door Openings

Measures should be initiated towards provision of peep holes and to closing door the during burner operation to arrest the heat losses. Implementation of this measure is expected to yield annual energy savings to the tune of Rs.2.680 lakhs/year. Refer Section 15.4.3 (E) for details.

Estimated savings in FO = 50.21 kL/year
Cost savings = Rs.2.680 lakhs/year
Investment required = Marginal
Simple payback period = Immediate

15.4.4 SUMMARY OF POTENTIAL SAVINGS

.Recommendation	FO Savings kL/yr	Cost savings Rs.lakhs	Investment required Rs.lakhs	Payback period (yrs)
Avoiding heat loss through door opening	50.21	2.68	00.00	Immediate
Total	50.21	2.68	0.00	Immediate

15.5.0 ROTARY FURNACE

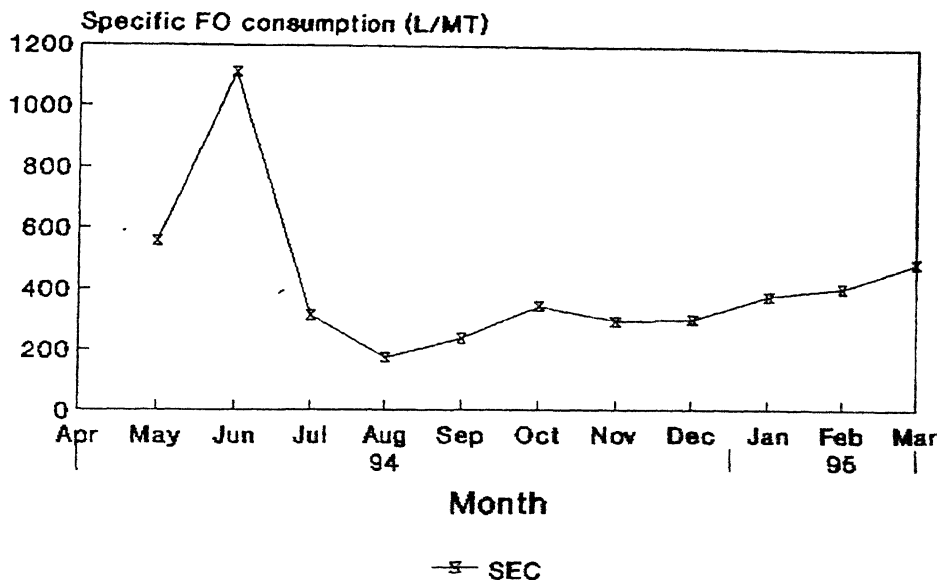
15.5.1 FACILITY DESCRIPTION

Plant has two rotary furnaces of 5 MT capacity out of which one will be in operation. The rotary furnace has a length of 3.2 m and diameter of 3 m made of mild steel shell. The furnace has an inner asbestos lining of 20 mm followed by a layer of insulation brick of 230 mm thick and a layer of magnochrome bricks of 230 mm thick. The burner is fixed from rear end uses furnace oil or LDO.

The antimonial lead in the refinery dross is recovered by heating the dross to a temperature of 900-1000°C. The molten lead is tapped and recycled to lead refinery. After lead tapping the remaining dross is converted into slag by heating upto 1100-1200°C and finally removed. Input materials to rotary furnace and the composition along with batch times are given in Appendix - 15.5/1.

15.5.2 ENERGY CONSUMPTION

The fuels used in rotary furnace are furnace oil and coke breeze. LDO is resorted only during the breakdowns of furnace oil line. The rotary furnace is provided with oil flowmeter and the consumption of fuel recorded every hour.

During Apr 1994- Mar 95, the unit has consumed about 256 kL of furnace oil and 25 kL of LDO. The corresponding antimony of lead production was 838 MT. Month-wise production and fuel consumption for the year 1994-95 alongwith specific energy consumption are given in Appendix - 15.5/2.

The average specific energy consumption (L of FO per MT of lead) is estimated at 333.848, while the minimum and maximum values are 173.913 and 1111.111 respectively. The variation in specific fuel consumption is due to wide variation in production.

15.5.3 OBSERVATIONS, ANALYSIS AND FINDINGS

A. Energy Balance

Efficiency of rotary furnace has been carried out by adopting indirect method in which various losses are deducted from total heat input to arrive at useful heat and efficiency. Various parameters such as flue gas temperature, %CO_2 in flue gas, surface temperatures of furnace, fuel consumption, combustion air flow rate were observed and recorded.

The average values of the above parameters are given below :

Sl No.	Parameters	Unit	Value
1	CO_2 in flue gas	%	3
2	Furnace temp.	°C	750
3	Oil flow rate	l/h	143
4	Combustion air velocity	m/s	12
5	Ambient dry bulb temp	°C	30
6	Ambient wet bulb temp.	°C	28

Estimation of various losses and thermal efficiency are given in Appendix - 15.5/3. Efficiency of the furnace has been worked out to be 44.94%.

Summary of heat losses and efficiency

Heat balance	kcal/h	Percentage
Heat Inputs		
Heat given through fuel	1385670	83.44
Heat given through coke	275000	16.56
Total	1660670	100.00
Heat output		
Useful heat	746164	44.94
Surface heat losses	130606	7.86
Flue gas losses	783900	47.20
Total	**1660670**	**100.00**

The useful heat consists of heat given to lead, heat given to slag and heat given to complex reactions in the furnace.

16.0 ZINC OXIDE PLANT

16.1 FACILITY DESCRIPTION

The residue produced from leaching plant (called moore cake) and blast furnace slag which contains zinc ferrite is treated in Zinc Oxide plant for recovery of zinc and lead. Residue contains about 16-20% zinc, 5-7% lead, and 0.1-0.2% cadmium. Zinc is present in the form of Zns, ZnO, $ZnSO_4$, $ZnO.Fe_2O_3$ and $ZnO.SiO_2$.

Zinc oxide plant comprises two sections :

a. Waelz kiln and
b. Clinker kiln

a. Waelz kiln

Waelz kiln consists of an inclined rotary kiln of 40m length and 3m diameter, dust chamber, tubular coolers and baghouse.

The kiln has four different zones., viz, zone-1 in which the feed is dried and preheated, zone-2 decomposition and reduction of compounds takes place, zone-3 is volatilization zone, and zone-4 is slag forming zone.

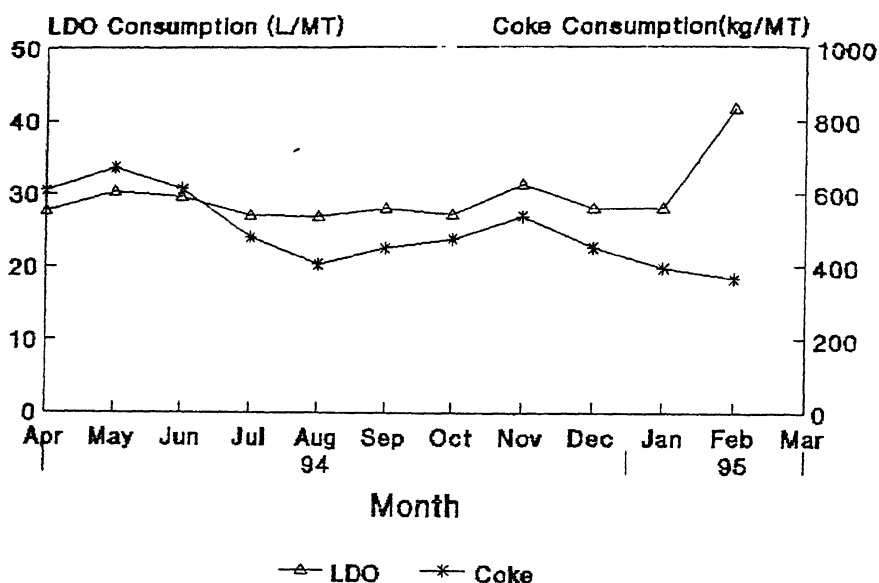
b. Clinker kiln

Clinker kiln is of 24 m length and 1.8 m dia having capacity of 1.5 MT/h. The basic objective of clinker kiln is to condition the raw zinc oxide for subsequent leaching operation by volatilizing chlorine and fluorine; lead and cadmium from zinc oxide. Raw zinc oxide of waelz kiln is fed to clinker kiln to produce clinker oxide of the following composition :

Zinc	=	50-55%
Lead	=	15-18%
Cadmium	=	0.09%
Chlorine	=	0.05%
Ferrous	=	0.0015%

16.2 ENERGY CONSUMPTION

During April 94 - February 95, the plant has consumed about 11466 MT of coke, 653 kL of LDO, 23006 MT of moore cake.

The month-wise specific energy consumption values of coke, LDO were estimated and tabulated in Appendix 16/1.

The average coke and LDO consumption per MT of moore cake processed is 498.392 kg and 28.384 L respectively.

16.3 OBSERVATIONS, ANALYSIS AND FINDINGS

A. Operating Parameters of Waelz Kiln

An attempt has been made to understand the dynamic operation of the kiln by systematic observations of the parameters such as kiln exhaust gas analysis, temperatures of kiln heads, slag temperature, feed rate, kiln speed, temperatures at the gas inlet and outlet of the dust chamber, bag filter and combustion air flow rate. Hourly observation of the above parameters have been tabulated in Appendix - 16/2.

B. Heat Balance of Waelz Kiln

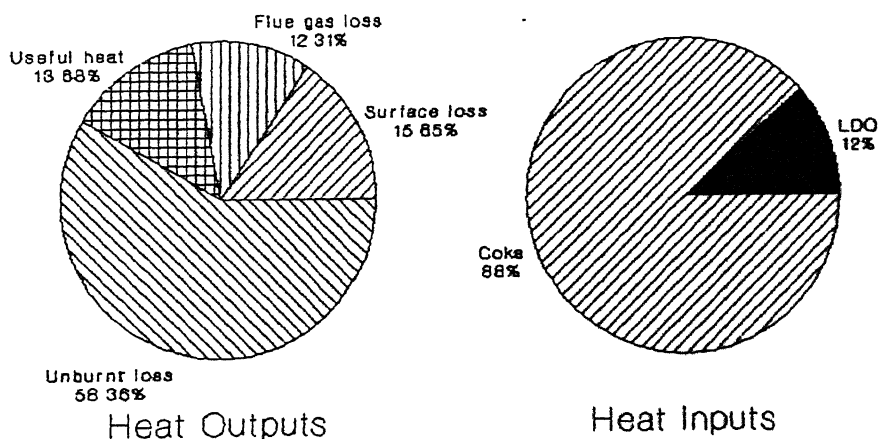
Heat balance of waelz kiln has been attempted at by quantifying various heat inputs and outputs to the kiln.

Heat Inputs

a. Heat value of LDO
b. Heat value of coke

Heat Output

a. Surface heat losses
b. Heat in exhaust flue gases
c. Heat loss due to unburnt carbon in slag
d. Heat of reaction, heat given to material and unaccounted losses.

Calculation of heat balance details are given in Appendix - 16/3. The following pie diagrams depicts percentages of heat inputs and outputs :

C. Use of FO in Place of LDO

The existing hourly cost of LDO consumption in Waelz kiln and clinker kiln worked out to Rs.797/- and Rs.344/- respectively.

Feasibility has been studied to replace existing LDO with furnace oil, considering the power consumption for furnace oil heating, It is estimated that the cost of hourly furnace oil consumption by proposed system is found to be Rs.577/- and Rs.295 in Waelz kiln and clinker kiln respectively.

From the above, it can be observed that FO usage is more economical in comparison with LDO. The complete techno-economics feasibility is given in Appendix 16/4. The table below gives the salient features of the feasibility.

Kiln	Cost of heating Rs./h		Saving Rs.lakh/yr	Eq.LDO saving MT/yr	Investment reqd. Rs.lakh
	Present	Proposed			
Waelz	797	577	15.28	209	7.8
Clinker	344	295	3.41	47	4.0
	1141	872	18.69	256	11.8

It can be seen that annual cost savings to the tune of Rs.18.69 lakh can be envisaged with an initial investment of Rs.11.80 lakhs.

D. Unburnt Carbon in Waelz Kiln Slag

Analysis of waelz kiln slag reveals that unburnt carbon present in slag is in the range of 30-35% Heat loss to the extent of 58.35% of total heat input is being lost due to unburnt in the slag. This carbon in slag can be recovered by installing slag crusher and magnetic separator in the slag outlet path. The plant personnel have already identified this measure and is under process of implementation.

E. Replacement of Pneumatic Conveying by Mechanical Conveying

The raw zinc oxide from tubular coolers of waelz kiln is received in hopper, from there it is pneumatically conveyed to the storage silo. The power required for pneumatic conveying is estimated at 13 kW.

Based on data provided by plant personnel, techno-economics was studied to replace the pneumatic conveying with mechanical conveying. The outcome of the analysis is as under :

Particulars	Unit	Pneumatic conveying	Mechanical conveying
Material transfer rate	MT/h	2	2
Power consumption	kW	13	7.5
Specific power consn.	kW/MT	6.5	3.75
Savings in specific power	kW/MT	-	2.75
Savings in power	kW	-	5.50

The annual energy savings are estimated at 27500 kWh amounting Rs.0.712 lakhs. The mechanical conveying requires two motors viz., horizontal and vertical. For horizontal transfer - screw conveyor and for vertical transfer, a bucket elevator can be used. The investment required would be in the order of 10.0 lakhs and payback period works out to 14.04 years. Since the payback period is more, further investigation is required to arrive at feasibility of the measure. The detailed calculations are given in Appendix - 16/5.

16.3 RECOMMENDATIONS

Use of Furnace Oil in Place of LDO

The cost of hourly heating in waelz kiln and clinker kiln should be reduced by using furnace oil in place of LDO. The cost savings are estimated at Rs.18.69 lakhs per annum with an investment of Rs.11.80 lakhs. Refer Section 16.3 C for details.

Estimated savings	= 256 kL/year
Cost savings	= Rs.18.69 lakhs/year
Investment required	= Rs.11.80 lakhs/year
Simple payback period	= 0.63 year

16.4 SUMMARY OF POTENTIAL SAVINGS

Recommendation	Energy Savings (kL/year)	Cost savings Rs.lakhs	Investment required Rs.lakhs	Payback period (yrs)
Using furnace oil instead of LDO	256	18.69	11.80	0.63
Total	256	18.69	11.80	0.63

17.0 CHILLING COMPRESSOR

17.1 FACILITY DESCRIPTION

The 50 TPD sulphuric acid plant is provided with a separate chilling compressor of rated capacity 4 x 10^3 kcal/hr to cool the SO_2 bearing gases as well as remove moisture. Salient features of the refrigeration unit is given below :

Sl. No.	Equipment Reference	Parameter
1.	Compressor	Rated HP - 180
2.	Condenser	Design water flow rate - 120 m^3/hr Total heat transfer area - 113.7 m^2
3.	Chiller	Design water flow rate - 80 m^3/hr Total heat transfer area - 74.6 m^2
4..	Chilled water pump	Design flow - 30 m^3/hr Design head - 30 MLC

17.2 OBSERVATIONS, ANALYSIS AND FINDINGS

A. Performance Assessment

Towards performance assessment, sample observations of various parameters such as chilled water inlet, outlet temperatures and pressures, discharge gas pressure, condenser water inlet & outlet pressure have been made for 6.8.95 & 7.8.95. Summary of observations have been give below :

Data	6.8.95	7.8.95
Water temperature drop across the gas cooler (°C)	6.0	5.0
Chiller exit gas pressure (psig)	244	240
Chilled water pressure drop across the chiller (kg/cm^2g)	1.2	-
Water pressure drop across the condenser (kg/cm^2g)	1.6	1.7

Chilled water temperature drop of 6 °C, pressure drop of 1.2 kg/cm^2g and gas discharge pressure of 240 psig indicate satisfactory operation of the machine. Condenser water pressure drop of 1.6 kg/cm^2g indicate the need to take up cleaning of the condenser.

18.0 DIESEL GENERATORS

18.1 FACILITY DESCRIPTION

Installed capacity of Diesel power house generators are as given below :

Sl. No.	No. of Sets	Rated Power Generation MW	Make
1.	3	5	Allen - NEI
2.	2	3	Russky Diesel Engine

Further details of the Generator Sets are given in Appendix - 18/1.

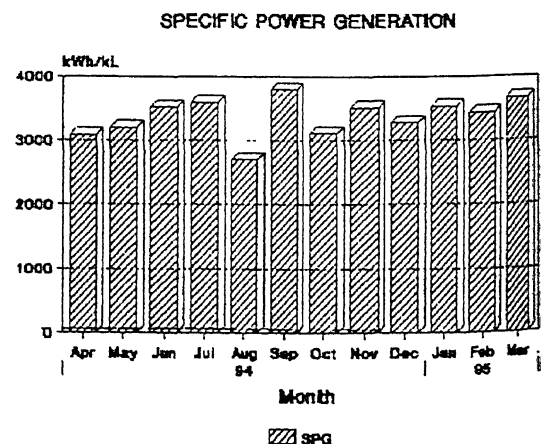
18.2 ENERGY CONSUMPTION PATTERN

The annual self-generated power for the year 1994-95 is as given below :

Generator Capacity	3 MW Set	5 MW Set	Total
Power generation	2351170	20395030	22746200
Percentage of total generation	10.3	89.7	100

It can be observed that 89.7% of the total self generated power of 227.462 lakh units is contributed by 5 MW sets.

The Exhibit besides shows the monthwise specific power generation per kL of HSD for the year 1994-95. HSD consumption during the year 1994-95 at a value of 6533.859 kL gives a specific power generation 3.48 kWh/litre.

Details have been given in Appendix - 18/2.

The annual running hours of the Diesel Generators for the year 1994-95 is as given below :

Generator capacity	3 MW Set	5 MW Set	Total
Annual running hours	1340.65	5593.6	6934.25
Percentage running hours	19.3	80.7	100

It can be observed that 5 MW sets run for the major part of the time (i,e, 80.7% of the total). DG sets-wise running hours are given in Appendix - 18/3.

18.3 OBSERVATIONS, ANALYSIS AND FINDINGS

A. Specific Power Generation Ratio

An observation was made on DG Set nos. 4 & 5 by monitoring fuel consumption and power generation. Specific power generation ratio have been worked out to be 3.06 kWh/litre and 3.145 kWh/litre respectively. Calculation details are given in Appendix - 18/4.

B. Performance Assessment

Towards assessment of effective working of DG sets, the following parameters have been monitored for set nos. 4 & 5.

i. Jacket water inlet & outlet temperatures

ii. Soft water and raw water inlet-outlet temperatures

iii. Cylinder-wise exhaust temperatures

Summary of observations is given below :

Details	Diesel Engine No.4	Diesel Engine No.5
Temperature drop across intercooler (°C)	6.0	-
Temperature drop across lube oil cooler (°C)	-	0.5
Temperature rise of water across J/W cooler (°C)	11.0	12.0
Temperature drop of air across intercooler (°C)		
`A' Bank	102.2	-
`B' Bank	99.2	64.0

C. Operation of DG Set for Part of Base Load Requirements and Waste Heat Recovery

Demand requirements of plant are met with APSEB and 100% demand has been utilised during power-cut free periods during 1994-95. Whenever demand restrictions or energy/demand cuts are imposed on the contract demand, DG sets are continuously run. It is understood that plant has approached APSEB for additional 3000 kVA of M.D.

From Appendix - 3/6, it is observed that peak load restrictions are imposed except during monsoon months. As such it is observed from records that one DG set is operated for 6900 hours during 1994-95.

Appendix - 18/2 gives the details of self generation cost and cost of APSEB electricity.

One 5 MW Allen Nei Ape DG Set may be operated continuously out of three DG sets as part of plant base load requirements. Details are given in Appendix - 18/6.

The above measure has been proposed and further discussions are made in subsequent chapter 18.3 D for waste heat recovery systems.

Implementation of above measure yields additional benefits to plant power system management as given below :

a. The maximum demand monitoring at 100 % load requirements can be eased and tripping of loads for demand control can be avoided (when power cut from APSEB is Nil).

b. The 33 kV bus power factor can be improved and system pf and instantaneous kVA demand can be at optimal values.

D. Potential Waste Heat Recovery from Diesel Generators

An attempt has been made to assess the heat recovery possible from exhaust gases of DG Sets. The potential steam generation can be to the extent of around 2500 kg/hr. This is suffice to meet the steam requirement of 300 TR of refrigeration loads in vapour absorption machine to be installed. This would be able to meet about 126 MT/hr of electrolyte cooling which is equivalent to one cooler load.

Implementation of the above measure yields annual energy savings (including demand charges) to the tune of about Rs.40,99,330. At an estimated investment of about Rs.110 lakhs, it works out to a simple payback of 2.7 years. Calculations details are given in Appendix - 18/7.

18.4 RECOMMENDATIONS

One DG Set of 5 MW capacity should be run continuously and waste heat recovery system/vapour absorption machine should be installed for obtaining 300 TR of refrigeration load. This would suffice to meet one cooler load of spent electrolyte system. The details and specific advantages in power system load management are given in section 18.3.C.

Implementation of the above measure is expected to yield energy savings as below :

Cost of annual energy savings = Rs.40.99 lakhs
Estimated budgetary investment = Rs.110.0 lakhs
Simple payback period = 2.7 years

Details are worked out in Appendix - 18/7.

18.5 SUMMARY OF POTENTIAL SAVINGS

Sl No	Proposal	Estimated Energy Savings		Cost savings Rs lakhs	Cost of implemen -tation Rs. lakhs	Simple payback period (years)
		Thermal kL/yr	Electrical (kVA M D)			
1	Waste heat recovery from DG Sets	-	3000	40 99	135	5 4
Total		-	3000	40.99	135	5.4

19.0 LIGHTING SYSTEM

19.1 FACILITY DESCRIPTION

The total connected lighting load is around 250 kW. The plant makes use of different kinds of fittings such as Incandescent, Fluorescent tubes, Mercury blended lamps, High pressure mercury vapor lamps (HPMV), High pressure sodium vapor lamps (HPSV) and Low pressure sodium vapor lamps (LPSV).

19.2 OBSERVATION, ANALYSIS AND FINDINGS

A. GENERAL

1. The details of distribution of light fittings in the plant is given in Appendix - 19/1.

2. Natural lighting has been extensively used but there is a significant number of artificial lights switched on during the day time.

3. The company has installed slim tube low pressure sodium vapor lamps for yard lighting to a large extent which is a very good energy conservation measure.

4. There is a changeover from fluorescent tubes to HPMV and HPSV lamps in the plant.

5. The general lighting levels measured in the various plants are given in Appendix - 19/2.

6. During the period of study it was observed that a number of lights in the yard on switched `ON' during the day. The list is given in Appendix 19/3. Care should be taken to switch off these lights.

B. Replacement by more efficient lighting

1. Incandescent lamps of 200 W and 500 W are used extensively in the plant. These lamps are most inefficient lamps and have a short life of about 1000 hrs only.

The table below gives details of the location of the incandescent lamps in the plant.

Sl. No.	Location	GLS	
		200 W	500 W
	BLAST FURNACE		
1.	Storage of material	1	1
	LEAD REFINERY		
2.	Ground floor kettle blower MCC	16	-
	GAS CLEANING		
3.	Cooling tower area	3	-
	CHARGE PREPARATION		
4.	Sinter preparation	4	-
	DL PLANT		
5.	DL M/c Area	2	-
6.	Hammer Mill	1	-
	CELL HOUSE		
7.	Outside electrolysis area	1	-
8.	Cell house towards road side	1	-
9.	Cathode charging furnace	-	1
10.	Pachuka Area - IV Floor	-	1
	COMPRESSOR HOUSE		
11.	Compressor House	-	1
	ACID PLANT		
12.	Acid plant area 200 TPD	-	2
	COOLING TOWER		
13.	Cooling tower periphery	-	1

As per plant, these are temporary lighting provided. However, it should be replaced with energy efficient lighting wherever possible.

2. There are 250 W and 400 W HPMV lamps which can be replaced by 150 W and 250 W HPSV respectively. The lumen output of 250 W and 400 W HPMV compares with that of 150 W and 250 W HPSV lamps respectively. A direct saving of 100 W and 150 W can be achieved by the use of HPSV lamps (150 W and 250 W) in place of 250 W and 400 W HPMV lamps respectively.

The table below gives the location of the 250 W and 400 W HPMV lamps which are to be replaced.

Sl No	Location	HPMV	
		250W	400W
	BLAST FURNACE		
1.	Periphery Ground floor	7	-
2	I Floor	4	-
3.	II Floor	2	-
4	III Floor	2	-
5.	Sinter Storage area	2	-
6.	Charging area Coke & Sinter	1	-
7.	Slag removal Area	1	-
8.	Rotary Furnace Area	2	-
	LEAD REFINERY		
9	Agitator area I Floor	2	-
	GAS CLEANING		
10	Cooling tower area	-	2
11	Gas cleaning blower area D C Motor	-	3
	DL PLANT		
12.	DL M/c Area	6	-
13	Charge preparation slag + stock	4	-
14	I Floor	14	-
15	Lead Mechanical	1	-
	CELL HOUSE		
16.	Outside electrolysis area	4	-
17.	Electrolyte cooler	3	-
18	Cell house towards rectifier side	-	2
19	Cell house towards road side	-	6
20	Cathode charging furnace	-	2
21	Ingot casting area	-	14

Sl. No	Location	HPMV	
		250W	400W
	M R S		
22	Transformer Yard	-	1
	WATER TREATMENT PLANT		
23.	Outside lighting	3	-
	LEACHING		
24	Leaching mechanical road	3	-
25.	Ball mill area - I Floor	-	3
26	Pachuca Area - IV Floor	-	5
27.	Main Bridge	3	-
28	Sand settler	-	3
29	Dorr thickener	-	2
30	Purification	-	11
31	Purification discharge pump	1	-
32	Pachuka discharge pump	2	-
	COMPRESSOR HOUSE		
33	Entrance	-	2
34	Compressor House	-	11
	ACID PLANT		
35.	50 TPD Cooler area	1	1
	TAIL GAS TREATMENT PLANT		
36	Tail Gas plant area	1	-
	COOLING TOWER		
37	Cooling tower periphery	1	-
	ROASTER PLANT		
38	Yard	3	-
39	D M water plant	-	2

Energy savings to the tune of 57960 kWh/year amounting to Rs.220248/- can be achieved with an investment of Rs.337000. The payback period is 1.53 years. The detailed calculations are given in Appendix 19/5.

3. The central workshop is lit by 400 W HPMV lamps. The measured lux levels in this shop is about 40 lux. The minimum required lux level in this shop is 200 lux. The number of lamps used in this workshop are 13. HID metal halide lamps can be used in this workshop which can be retrofitted to the existing HPMV luminaire with the addition of an ignitor. These lamps simulate daylight appearance and have better spread of light. The number of luminaires required to maintain a lux level of 200 is 9. Energy savings to the tune of 7680 kWh/year amounting to Rs.29184/- can be achieved with an investment of Rs.27000/-. The payback period is < 1 year. Detailed calculations are given in Appendix 19/6.

C. Voltage Controllers for Lighting System

Energy savers or voltage controllers are not installed in the lighting system. These energy savers will save about 15% of the energy consumed by the lighting circuit. This energy saver/voltage controllers reduces the power as the square of the voltage without impairing the ability of the luminaire to strike. The reduction in the power supplied to discharge lights can be made without a proportional drop in the light output. Energy savings to the tune of 93366 kWh/year amounting to Rs.354790/- can be achieved with an investment of Rs.419900/-. The payback period is 1.18 years. Refer Appendix 19/7 for detailed calculations.

19.3 RECOMMENDATIONS

A. General

The yard lights should be put off without fail during the day time and care should also be taken to switch off the lights at the workplace also.

B. Replacement by more efficient lighting

1. The 250 W and 400 W HPMV lamps should be replaced by 150 W and 250 W HPSV lamps. Energy savings to the tune of 62280 kWh/year amounting to Rs.158190 can be achieved. Refer Section 19.2 B.2 for details.

   Energy savings = 57960 kWh/year
   Cost savings = Rs.220248/year
   Cost of implementation = Rs.337000/-
   Simple payback period = 1.53 years

2. HPMV lamps in the central workshop should be replaced with HID. Metal halide lamps which will save energy of 7680 kWh/year amounting to Rs.20250. Refer Section 19.2 B.3 for details.

   Energy savings = 7680 kWh/year
   Cost savings = Rs.29184/year
   Cost of implementation = Rs.27000/-
   Simple payback period = < 1 year

C. Voltage controllers for lighting system

Voltage controllers/Energy savers should be installed in the lighting system which will save energy to the tune of 93366 kWh/year amounting to Rs.354790. Refer Section 19.2 C for details.

Energy savings = 93366 kWh/year
Cost savings = Rs.354790/year
Cost of implementation = Rs.419900/-
Simple payback period = 1.29 years

1.9.4 SUMMARY OF POTENTIAL SAVINGS

Sl No	Proposal	Savings		Cost of implementation (Rs)	Simple payback period Years
		Energy kWh/year	Cost Rs /year		
1	Replacement of 250 W and 400 W HPMV by 150 W and 250 W HPSV Lamps respectively	57960	220248	337000	1 57
2	Replacement of 250 W HPMV by 250 W HID Metal Halide Lamp	7680	29184	27000	< 1 year
3	Voltage controllers for lighting system	93366	354790	419900	1 18
Total		159006	604222	783900	1 29

20.0 CADMIUM PLANT

20.1 FACILITY DESCRIPTION

Cadmium is an important by-product of Zinc-hydrometallurgy. Ist stage purification cake of Leaching and Purification plant contains 4-8% of cadmium, 1-3% copper and 40-50% of zinc. The leached solution of the above cake analyses cd = 14-20 gpl ; Zn = 130 - 160 gpl. The leached solution on reaction with zinc dust cements out cadmium. Cadmium sponge is leached with conc. sulphuric acid and purified. The purified solution is mixed with cadmium spent electrolyte and fed to cells. There are 10 electrolytic cells as per the details given below :

i. 22 nos. of Aluminium cathodes
ii. 23 nos. of Lead anodes
iii. Current density 40 A/M^2

The cathodes are stripped once in 24 hours. The cathodes are washed, dried and melted in electric resistance furnace and cast into pencils.

20.2 ENERGY CONSUMPTION PATTERN

Month-wise cadmium production and power consumption for the period May-Oct 94 is given in Appendix - 20/1. Electrolytic cells constitute major power consuming area. Average monthly power consumption is in the range 12000 kWh - 14000 kWh. Average specific power consumption for the above period works out to 1.97 kWh/MT.

20.3 OBSERVATIONS, ANALYSIS AND FINDINGS

Total measured cell voltage for 10 nos of cells works out to 23.5 V. The average anodic millivolt drops comes to 4.5 mV. The above values are within permissible limits.

21.0 ENERGY MANAGEMENT - AN OUTLOOK

21.1 INTRODUCTION

The energy bill of the unit runs to about 20 % of the total manufacturing cost, which will continue to escalate with the inescapable raise in cost of supply of electricity and fuels, in the coming years.

In order to control the excessive consumption of energy and bring maximum possible savings in energy consumption, it is essential that an effective energy management system and process is initiated in the organisation.

Energy Management requires a logical and comprehensive management approach. Energy savings become significant and long lasting when they are achieved as part of an overall plant energy management programme. A systematic and structural approach is essential to identify and to realise the full potential savings.

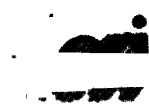
The most essential requirement for a successful energy management programme is the top management commitment. An important part of top management commitment is to create an organisation for implementing the energy management programme. This is commonly at two levels, the Energy Manager and the Energy Committee. Evidence of top management commitment will be seen in the level of support given to the Manager and the Committee, in all respects.

The basic requirements for the position and the job description of a typical Energy Manager are described at the end of this annexure.

21.2 MANAGEMENT APPROACH

A. Top Management Commitment

The most essential requirement for successful energy management programme is the commitment of top management. They must visibly demonstrate their commitment to the employees of the enterprises.

The decision of the company to control energy costs must be clearly stated and understood by all within the company. Senior management should participate in energy related activities. The company Chief Executive should regularly call for information/reports on the progress, particularly at the beginning of the energy management programme.

B. PRELIMINARY ANALYSIS

In order to develop a energy management programme in the proper perspective, it is necessary that the scope, extent of detail and the management cost and time expended should have some relation to the potential benefits realisable by the programme. There is no point if the cost incurred is more than the value of energy saved.

The energy management programme should begin with the analysis like -

- Consumption of different forms of energy
- Energy cost as a percent of total production cost
- Major energy consuming equipments and their diversity
- Potential savings and its comparison with current profits
- Estimate of costs of additional metering, that may be required
- Within the existing company organisation how best can energy consumption be monitored in different areas or departments

Such broad assessment would give a perspective of the management time and cost value in relation to potential returns.

C. ENERGY COMMITTEE

In manufacturing industry, close co-ordination with different functions will be essential. To achieve this co-ordination at larger manufacturing sites, an Energy Committee will be needed. This may, for example, include senior managers, the Accountant and the Chief Engineer. The Chairman should be the General Manager who has sufficient authority to ensure that all necessary resources are made available and any necessary action is taken.

The Energy Committee will be responsible for :

- developing the energy efficiency policy
- managing the monitoring system
- agreeing and reviewing standards and targets
- examining energy cost-saving schemes and ensuring projects are implemented
- other important matters concerning energy

Once a corporate decision has been made to initiate an energy management programme, a management structure within the company's organisational framework needs to be created, in view of the special role of energy as a common input across different divisions, departments and sections.

The energy management structure will depend on the size of the enterprises, its functional organisation, and its manufacturing activities.

D. ENERGY MANAGER

Looking at the size of energy bills, it appears essential that a full-time energy manager is appointed to implement the energy management programme. The appointment of an Energy Manager would also demonstrate to the company employees, the management commitment and its seriousness in dealing with the problem.

The Energy Manager should be appointed from within the plant, to ensure that he has good practical knowledge of all aspects of operations, both technical and administrative.

E. RESPONSIBILITY FOR RESULTS

In general, most important structures in manufacturing industry will be based in three levels of authority with corresponding responsibilities for the efficiency of energy use.

Level 1 : Senior Management : With responsibility for the efficiency with which energy is used in the organisation as a whole, in relation to other resources, and in the production of particular products.

Level 2 : Middle Management : With similar responsibilities for the efficient use in relation to specific areas of the manufacturing process or divisions of the organisation.

Level 3 : Process Operators, Foremen and Supervisors : With responsibility for maintaining control over the efficiency of energy use in a particular item of plant or part of a process.

At all three levels, those responsible for controlling and improving the efficiency of energy use will need regular reports on energy use in relation to standards and targets.

Providing these reports, analysing the energy data, developing standards of performance, and deriving the information needed for setting targets will be task of an Energy Manager who is responsible to the Energy Committee. His duties may also include responsibility for the installation and operation of metering systems and the training of staff responsible for the collection and analysis of energy data.

21.3 ENERGY MANAGEMENT PROCESS/STRATEGY

There are four distinct stages :

- Defining energy accounting centres
- Measurement
- Analysis & Monitoring
- Targeting

A. ENERGY ACCOUNTING CENTRES (EAC)

The first step in installing an energy management programme is to identify along the energy flow paths of the plant, a series of `Energy Accounting Centres' which will provide the requisite breakdown and frame-work necessary, both for monitoring energy performance and for achieving targets. An Energy Accountable Centre might consist of individual equipments a section/dept. or even a whole building.

Each centre must relate to a nominated individual responsible for operational achievement in that area. Tying resource consumption to those responsible for operational achievement is a key factor in energy management programme, since it focuses attention of those with the authority to bring about improvements in performance.

Those held accountable for energy performance should also be able to assess that performance and also have the pertinent information on which to base judgements, decisions and actions to bring about improvements.

Each energy accounting centre (EAC) requires the facility of meters, to measure the energy consumed over a period and a means of measuring/assessing the production (or other specific variable) over the same period. As far as possible the EAC's identified should correspond with existing cost control centres on the site.

B. MEASUREMENT

Before any resource can be managed effectively, it must be measured correctly in order to provide the information upon which to base management decisions. So, like all truly effective management systems, energy management depends on the collection of relevant data upon which to judge current performance and to plan for future improvements. The gathering of this information forms an essential part of the monitoring programme.

C. ANALYSIS & MONITORING

After collection of energy consumption and cost data, the next stage is to use that information to analyse and evaluate performance.

Analysis and evaluation involve, regularly comparing actual levels of energy consumption with the amount of energy expected to be used as defined by a set of internally based standards. Difference between actual consumption and these standards will reveal either improvements in energy efficiency or a fall-off in performance levels. In this way, the information produced by monitoring forms a basis for continuing performance evaluation and control.

On the one hand, it will provide quantified evidence of exactly how successful have been the measures to improve performance. On the other, it will indicate if and where failures have occurred and trigger the necessary remedial action.

Analysis should be a continuing process so that action can be taken speedily if energy efficiency deteriorates. And to ensure effective performance evaluation and control, each line manager or plant operator must receive the energy throughput and other figures regularly - on a weekly/ monthly basis - and promptly, so that departures from the standards can be quickly detected and corrected. In turn, line managers themselves must ensure that they respond rapidly to the information they receive. And here, well designed reporting forms, expressed in readily understood energy cost terms, will be very helpful.

Achieving greater energy efficiency depends on developing an energy management strategy that will maintain progressive reductions in energy consumption for the same or high levels of output. And the foundation of effective energy management is the introduction of a system of monitoring to equip the managements with the information and the motivation to attain greater levels of energy efficiency.

The essence of monitoring is that energy use is accurately measured, then compared with a set of standards derived from a knowledge of the organisation's own capability, and then possibly further checked by reference to external norms.

By wielding the control and motivational aspects of energy management closely, monitoring provides a structured framework in which managers at all levels are able to optimise efficiency through the careful use of the energy resources for which they are responsible.

Just by the introduction of a monitoring system alone, many organisations have found that they can cut their energy consumption by up to ten percent.

D. TARGETING

The first stage in the process of setting targets is to carry out an energy audit - a procedure which can with advantage be repeated every year.

An energy audit will identify the possible range of energy efficiency improvement measures available and appropriate to the circumstances of an individual organisation. It will also provide an estimate of their costs and the likely return on investment.

From the results of the audit, management can select a series of measures to form an action programme - starting with the most cost-effective and taking into account, for example, the availability of capital and effect of the measures on the organisations other activities.

In the first instance, the action programme may simply involve changing working practices or adjusting machinery. It may then move on through low cost improvements, like plant and pipework insulation, to investment in higher cost measures, such as heat recovery equipment or more energy-efficient plant.

Targets are then set for the implementation of change and the achievement of the predicted energy cost savings. The choice of targets will take account of current standards and the timescale for implementing measures. And an organisation may wish to set a range of targets, taking account of the scope for improvement, the resources allowed by management to effect improvement and the need to match accountability to the energy accountable centres.

There are two principal methods of target setting. In the first place, the so called `top down' approach, a broad based generalised technique which does not draw on a detailed analysis of the circumstances of the organisation but may be based on experience in the sector as a whole. In the second place, the `bottom up' which is based on a close knowledge of the energy requirements of different parts of an organisation's activities.

Both systems have their merits and which one is chosen depends on circumstances and cost-effectiveness. Experience has shown, however, the most organisations prefer the `bottom up' approach since it is, by its very nature, more closely tailored to their business needs and hence more effective in providing motivation.

Correctly set, targets have a strong motivational effect on the workforce. But it is important to avoid either impossible or too easy obtainable targets since these can be counter productive.

21.4 IMPORTANCE OF HUMAN ELEMENT

In the implementation of E.M. Process getting the human element right is vital to the success, like in any management system. So when introducing energy management into an organisation, it is essential to put people first of all, to establish a chain of managerial responsibility which reaches right up to senior management and which can motivate for improvements in energy efficiency throughout the organisation.

21.4.1 WAYS FOR FULLER CO-OPERATION OF PESONNEL

A. Education

A well thought-out familiarisation programme should convince employees of the need for good standards of housekeeping and energy awareness. They should appreciate, that it is in their best interests that all unnecessary and excessive use of energy be eliminated.

Energy cost savings add directly to profit. They will help safeguard, the employees' future by improving the firms economic well-being and competitiveness. Moreover, each rupee saved is equivalent to many rupees worth of extra production. It is important to emphasise that sacrifices are not being sought, nor are the staff being expected to work in less than satisfactory conditions.

Early encouraging results are unlikely to be sustained indefinitely. People do tend to drift back into their former habits, but the right climate of opinion will be established for introducing more complex, and lasting measures in a gradual manner.

B. Awareness and Information Sharing

In most plants, employees have little or no idea of the amount of energy being consumed within their plant, their section and even the equipment being operated by them. In such a situation, energy conservation obviously carries no meaning. Employees can be stimulated to support energy management by making them aware of the amount of energy they are using, the associated costs, the many ways to save energy, and the importance of energy conservation for the company's viability/profitability.

The information can be provided in the form of comparisons of historical trends, goals for overall energy use, energy intensity, etc., in both physical and monetary terms ; energy conservation checklists for each manufacturing operation; outlining simple and routine housekeeping measures to save energy; audio-visual presentations, and other literature.

Information must be presented in a manner which facilitates comprehension. If the information is too technical, too much theory, too sketchy, or too dull, it is likely to be ignored or not understood.

Terms that employees can relate to in everyday life should be used. For example, a sign saying " stop steam leaks" will not be as effective as a sign saying " A quarter inch diameter steam leak costs Rs.30,000 per month".

Training is also an important means of both informing and involving people at all levels in an energy management programme.

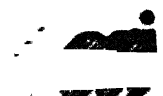
For operating personnel, training is required in practicalities of energy saving. This could be integrated into the organisation's other training programmes.

Upper level management also need to be informed of the overall energy situation, energy costs in relation to other costs, the energy management programmes - its goal, achievements, technical, economic and behavioral aspects etc.

C. **Motivation**

Motivation is based on involvement and commitment, and a sense of personal accountability can be generated only through total involvement of plant personnel at all stages.

Involvement must begin with the top management. As mentioned earlier, top management must be fully committed to the energy management programme and must visibly demonstrate their commitment and involvement in every manner possible and at every available opportunity. Top management must originate the programme, generate momentum and then maintain momentum. Adequate personnel and financial resources must be provided and responsibilities delegated to implement activities and projects to achieve the predetermined energy conservation goals. Progress should be monitored with goals reviewed and revised in the best interests of the company.

Operators and maintenance staff should be involved actively, as they are ultimately responsible for execution of activities in the programme. Also, they are often in a better position to recommend areas for savings or improvements. The most effective way of involving them is by simply going out and talking to them regarding goals, achievements, problems and progress or lack of progress. This demonstrates to them that the energy conservation programme is real and also that their role is important in success or failure of the programme.

Supervisors and middle level management should be involved by assigning them responsibilities for implementing and monitoring activities and submitting performance reports to top management, and by getting them to interact and communicate with operators and maintenance staff on progress and problems. If possible, energy management activities should be made a part of each supervisor's performance or job standard.

D. Incentives and rewards

Another method of motivating people is through incentives and rewards. Monetary rewards could be given to employees for suggestions leading to substantial energy savings ; for innovative ideas or solutions ; and for outstanding efforts in implementation of energy conservation activities. Wide publicity of effective idea provide an added incentive in the form of public recognition. Other incentives could be designed to meet the needs and attitudes of plant personnel.

E. Publicity

Publicity and promotion are essential to create climate for the energy management programme. Some commonly used means for publicizing and promoting energy conservation programmes are :-

1. Energy conservation performance results for plant and department, posted, monthly by the plant energy manager.

2. One article per month written in the company or plant paper or one good energy conservation idea that was implemented.

3. Articles from the company or plant paper used to obtain local newspaper interest and coverage.

4. Posters and pamphlets on energy conservation.

5. Letterheads with different energy conservation messages and ideas printed.

6. Plant-wide, high-visibility vehicles or equipment are used to carry signs publicizing energy conservation.

7. Plant energy manager having face-to-face energy conservation discussions with plant personnel. The opportunity checklist can be used for discussion topics.

8. Unit representatives and several unit personnel conduct quarterly on site reviews, a walkthrough of the unit looking for energy saving opportunities.

9. An agenda item on energy conservation included at staff meetings.

10. Energy conservation material provided to first-line supervisors for employee discussion.

11. Quarterly meetings held in the plant for all unit representatives.

12. An Energy Awareness Day is set aside in the plant twice a year.

13. A company energy logo developed and adopted.

21.5 KEY TASKS OF ENERGY MANAGEMENT

(1) Energy Data Collection and Analysis

* Record maintenance of all energy consumption in the plant.

* check the reading of all meters and submeters on a regular basis.

* specify additional meters required to provide additional monitoring capability.

* develop indices for specific energy consumption relative to production and maintain these indices on a monthly basis for all major production areas.

* set performance standards for efficient operation of machinery and facilities.

(2) Energy Purchasing Supervision

* review all monthly utility and fuel bills ; ensure billing is proper and that the optimum tariff is applied in each case.

* investigate and recommend fuel switching over opportunities where a cost advantage to the company is possible.

* develop contingency plans to implement in the event of supply interruptions or shortages.

* work with individual departments to prepare annual energy cost budgets.

(3) Energy Conservation Project Evaluation

* develop energy conservation ideas and projects, working with in-house staff, equipment vendors and outside consultants.

* summarise and evaluate possible energy saving projects according to the company financial planning requirements ; perform the necessary economic analyses to permit management evaluation of the projects.

* obtain management commitment of funds to implement conservation projects.

* re-evaluate possible projects as the company operations change or grow ; evaluate energy efficiency of new construction, building expansion or new equipment purchases.

(4) Energy Project Implementation

* initiate equipment maintenance programmes for energy saving

* supervise the implementation of conservation projects, including specification of equipment, requests for quotation, evaluation of offers, ordering of materials, construction/installation, operator training, start-up and final acceptance.

(5) Communications and Public Relations

* prepare monthly reports to management, summarizing monthly energy costs and consumptions as well as specific energy consumptions.

* communicate with all production and support departments, so that all participate in the energy management programme.

* develop an awareness programme within the company to encourage active participation by all employees in energy saving activities.

* develop training programmes to upgrade knowledge and skills of all levels of employees in energy related matters.

* publicise the company commitment to energy conservation where appropriate, providing information for press releases and internal notices, presenting papers in professional conferences, and entering the company in energy award programmes.

21.6 CHECKLIST FOR TOP MANAGEMENT

A. Inform line supervisors of :

1. The economic reasons for the need to conserve energy
2. Their responsibility for implementing energy saving actions in the areas of their accountability.

B. Establish a committee having the responsibility for formulating and conducting an energy conservation programme and consisting of :

1. Representatives from each department in the plant
2. A co-ordinator appointed by and reporting to management.

C. Provide the committee with guidelines as to what is expected of them :

1. Plan and participate in energy saving surveys.
2. Develop uniform record keeping, reporting and energy accounting.
3. Research and develop ideas on ways to save energy.
4. Communicate these ideas and suggestions.
5. Suggest tough, but achievable, goals for energy saving.
6. Develop ideas and plans for enlisting employee support and participation.
7. Plan and conduct a continuing program of activities to stimulate interest in energy conservation efforts.

D. Set goals in energy saving :

1. A preliminary goal at the start of the programme.

2. Later, a revised goal based on savings potential estimated from results of surveys.

E. Employ external assistance in surveying the plant and making recommendations, if necessary.

F. Communicate periodically to employees regarding management's emphasis on energy conservation action and report on progress.

DUTIES AND RESPONSIBILITES OF ENERGY MANAGER/CO-ORDINATOR

* To generate interest in energy conservation and sustain the interest with new ideas and activities.

* To maintain summaries of energy purchases, stocks and consumption, and to review and report on energy utilisation regularly.

* To be the focal point for departmental records of energy use, and to ensure that the records and accounting systems are uniform and in consistent units.

* To co-ordinate the efforts of all energy users and to set challenging but realistic targets for improvements.

* To give technical advice on energy-saving equipment and techniques, or to identify suitable sources of sound technical guidance on specialised subjects.

* To identify areas of plant activity which require detailed study and to give priority to such activities.

* To maintain records of all-indepth studies and to review progress.

* To provide a basic handbook of good energy practice for the plant operating department.

* To give specialist advice to purchasing, planning, production and the other functions of all aspects of energy conservation, especially on the long term implications.

* To ensure that, in making improvement in energy efficiency, health and safety are not adversely affected.

* To liaise with committees and working groups within his own industry, and provided no confidential data are involved, to exchange ideas on cost cutting techniques and performance figures for similar processes.

* To maintain contacts with research organisations, equipment manufacturers and professional bodies to ensure that he is up-to-date on significant developments in the field of energy conservation.

* To remain up-to-date on national energy matters and to advise senior company management on such topics, as well as co-operating with government departments in energy-related matters.

22.0 CONCLUSION

The scope for energy conservation in M/s Hindustan Zinc Limited, Visakhapatnam has been studied and discussed in detail. It is observed that there is a potential for energy savings by implementing the various measures suggested. It may be observed that the simple payback period worked out for each recommendation in most of the cases is less than three years. The list of retrofits and equipments along with their suppliers are given in Appendix - 22/1.

ACKNOWLEDGEMENT

We wish to place on record our grateful thanks to Shri P C Jain, General Manager, Shri B N Mittal, General Manager, Shri M C Gupta, Chief Manager (Maint.), Mr. V.K.B. Tally, Chief Manager and Shri Y B Narayana, Sr.Manager (Elec.) for all the co-operation extended to us during the study. We are also greatly indebted to all other officers and staff for extending necessary help.

APPENDICES

APPENDIX - 2/1

INSTALLED CAPACITY AND PRODUCTION DETAILS

Sl No	Product	Units	Installed Plant Capacity	Production		
				1994-95	1993-94	1992-93
1	Zinc Ingot	MT	30000	27025	30040 0	29702 5
2	Lead Ingot	MT	22000	11003	2415 00	16532 0
3	H_2SO_4	MT	75000	49472	56372 0	629262 53
4	H_2SO4 (Black)	MT		2565	1156 0	-
5	Cadmium	kg	115000	53 400	70860 0	113600 00
6	Silver	kg	30000	15 117	11191 0	4912 84

APPENDIX - 2/2

ENERGY CONSUMPTION DETAILS

ELECTRICITY CONSUMPTION

Year	Electy. Purchased (L.kWh)	Electy. Generated (L.kWh)	Total consmn. (L.kWh)
1992-93	1082.22	486.74	1569.01
1993-94	1233.16	263.64	1496.80
1994-95	1193.70	227.46	1421.16

ZINC PLANT

Year	LDO (kL)	FO (kL)	LSHS (kL)	LPG (kg)	Hard Coke (MT)	HSD (kL)
1992-93	2066.25	763.00	-	-	-	-
1993-94	2596.00	158.00	-	-	-	-
1994-95	2780.5	426.95	-	-	-	-

LEAD PLANT

Year	LDO (kL)	FO (kL)	LSHS (kL)	LPG (kg)	Hard Coke (MT)	HSD (kL)
1992-93	818.04	1205.783	-	-	9654.11	-
1993-94	840.70	1176.096	-	-	1646.60	11.602
1994-95	1450.95	196.80	351.43	18127	5074.595	-

APPENDIX - 2/3

COST ELEMENTS IN ZINC & LEAD PRODUCTION

Sl. No.	Cost Element	Zinc		Lead	
		1992-93	1994-95	1992-93	1994-95
1.	Feed material	54.8%	46.2%	60.0%	52.1%
2.	Chemicals	4.7%	4.5%	15.2%	15.9%
3.	Power	22.9%	28.9%	3.3#	5.0%
4.	Fixed cost	17.7%	20.4%	21.5%	25.5%
5.	Others	-	-	-	1.5%

ENERGY COST OF VARIOUS FUELS

Year	LDO (Rs /kL)	FO (Rs /kL)	Hard Coke (Rs /MT)	LSHS (Rs /kL)	LPG (Rs /kg)	HSD (Rs /kL)	Purchased Electy (Rs /kWh)
1992-93	5873 06	4946 0	3596 0	4246 0	-	5753 5	1.85
1993-94	6495 00	5324 00	4198 0	-	11 00	6801 0	2 06
1994-95	7310 00	5344 0	3115 0	-	10 80	7728 1	2 24

APPENDIX - 3/1

MONTHLY ELECTRICITY CONSUMPTION FIGURES FOR THE YEAR 1993-95

All kWh figures are L.kWh

Month/ year	Total		APSEB		DG		Rectifiers		Other Plant	
	kWh	MW	kWh	MW	kWh	MW	kWh	MW	kWh	MW
Jan 93	145.01	19.49	120.11	16.14	24.89	3.34	103.07	13.85	41.93	5.63
Feb	129.90	99.33	73.24	10.89	56.65	8.43	92.59	13.77	37.30	5.55
Mar	143.85	19.33	84.44	11.35	59.40	7.98	104.26	14.01	39.59	5.32
Apr	132.73	18.43	75.77	10.52	56.95	7.91	98.39	13.66	34.33	4.76
May	128.04	17.21	82.19	11.04	45.84	6.16	98.93	13.29	29.10	3.91
Jun	125.52	16.73	77.53	10.76	42.98	5.97	92.47	12.84	28.04	3.89
Jul	58.41	7.85	53.69	7.21	4.72	0.63	37.59	5.05	20.82	2.79
Aug	125.33	16.84	106.72	14.34	18.61	2.50	97.46	13.10	27.87	3.74
Sep	131.59	18.27	131.78	18.30	0.19	0.03	94.76	13.16	37.02	5.14
Oct	132.08	17.75	132.03	17.74	0.54	0.01	96.56	12.98	35.52	4.77
Nov	134.19	18.63	132.10	18.34	2.09	0.29	92.28	12.81	41.91	5.82
Dec	144.71	19.45	140.70	18.91	4.01	0.54	103.55	13.91	41.16	5.53
Jan 94	142.93	19.21	136.52	18.35	6.41	0.86	102.10	13.74	40.82	5.48
Feb	117.23	17.44	90.66	13.49	26.57	3.95	90.93	13.68	25.30	3.76
Mar	128.79	17.31	73.61	9.89	55.16	7.41	98.32	13.20	30.45	4.09
Apr	31.87	4.43	24.31	3.37	7.55	1.05	9.01	12.52	2.29	3.18
May	75.37	10.13	50.76	6.80	24.60	3.30	49.37	6.63	26.18	3.52
Jun	93.90	13.04	52.18	7.24	41.72	5.79	68.46	9.51	25.44	3.53
Jul	124.70	16.76	98.95	13.30	25.75	3.46	91.58	12.31	33.12	4.45

Appendix - 3/1 contd..

Month/ year	Total		APSEB		DG		Rectifiers		Other Plant	
	kWh	MW	kWh	MW	kWh	MW	kWh	MW	kWh	MW
Aug 94	141.87	19.07	141.40	19.00	0.46	0.06	101.27	13.61	40.60	5.46
Sep	131.44	18.25	129.91	18.04	1.52	0.21	89.45	12.42	41.99	5.83
Oct	142.82	19.19	142.23	19.11	0.59	0.08	100.78	13.54	42.04	5.65
Nov	135.51	18.82	132.04	18.39	3.46	0.48	93.88	13.04	41.62	5.78
Dec	136.33	18.32	135.31	18.18	1.01	0.13	95.37	12.81	40.96	5.50
Jan 95	144.76	19.45	128.27	17.24	16.43	2.20	102.75	13.81	42.00	5.64
Feb	129.00	19.19	79.25	11.79	49.74	7.40	93.90	13.97	35.09	5.22
Mar	133.39	17.92	79.03	10.62	54.35	7.30	98.95	13.30	34.43	4.62
Apr	127.29	17.68	76.30	10.59	50.94	7.07	91.48	12.70	35.81	4.92
May	73.42	9.86	53.94	7.95	19.48	2.61	46.25	6.21	27.17	3.65
Jun	114.45	15.89	68.06	10.84	36.38	5.05	92.04	12.78	22.43	3.11

APPENDIX - 3/2

NAME PLATE DETAILS OF POWER TRANSFORMERS

	Incoming	Outgoing
Transformer No.	1 & 2	X11 & X12
Transformer rating	35/40 MVA	10/12.5 MVA
Manufacturers name	NGEF	Bharat Bijilee
Year of Manufacture	1990	1975
Type of transformer and Vector group	OFTR-40000/132E YNYno	- Dy11 +30° group 4
Type of cooling	ONAN/OFAF	ONAF/OFAF
Rated pri/secy volt	132/33 kV	33/6.9 kV
Rated pri/secy amp	131.2-153 --------- 524.8-612.3	175-836 ------- 218-1014
% Impedance (base)	12.12 (35 MVA)	9.3 - 11.66 ----------- 9.21-11.49
Oil quantity	19205 L	7340 L
Oil mass	16.9 t	5300 kg
Untanking mass	36 t	10850 kg
Core & winding	16.1 t	7340 kg
Total weight	69 t	23490 kg
No.of taps	17	-
Tap changing	Auto	Auto
OLTC range	118.8 kV to 145.2	29.7 kV to 36.3

APPENDIX - 3/3

NAME PLATE DETAILS OF GENERATOR TRANSFORMERS

Transformer No.	3 & 4	5
Transformer rating	10/12.5 MVA	10/12.5 MVA
Manufacturers name	GEC	Bharat Bijilee
Year of Manufacture	1989	1991
Type of transformer and Vector group	YND11	YNd11
Type of cooling	ONAN/OFAF	ONAN/OFAF
Rated pri/secy volt	11/33 kV	11/33 kV
Rated pri/secy ampere	131.2-175 --------- 393.6-525	131.5-175 -------- 394-525
% Impedance	7.364-9.818 7.091-9.456	8.96/9.51
Oil quantity l	4250	4350
Oil mass kg	3700	3741
Untanking mass kg	8400	8600
Core & winding kg	9900	4109
Total weight kg	22000	17750
No.of taps	7	7
Tap changing	Manual	Manual

APPENDIX - 3/1

NAME PLATE DETAILS OF 6.6/0.433 kV TRANSFORMERS

Rated capacity		=	1600 kVA
No.of transformers		=	12 Nos.
Phase		=	3
Frequency		=	50 Hz
Cooling		=	ONAN
Volts at No Load	HV	=	6600
	LV	=	433
Full load Amperes	HV	=	140
	LV	=	2135
Standard used		=	IS:2025/1962
Connection		=	Delta/Star
Rated temp. rise in oil		=	45°C
Rated temp. rise by resistance		=	55°C
Vector group ref No.		=	DY11
Qty of oil		=	1650 l

1000 kVA Transformers (2 Nos.)

Rated capacity		=	1000 kVA
No.of transformers		=	2 Nos.
Phase		=	3
Frequency		=	50 Hz
Cooling		=	ON
Volt at No Load	HV	=	6600
	LV	=	433
Full load Amperes	HV	=	87.5
	LV	=	1332
Standard used		=	IS:2026/1962

Rated temp. rise in oil	=	45°C
Rated temp. rise by resistance	=	55°C
Vector group ref No.	=	Dy11
Qty of oil	=	1150 l

1250 kVA Transformers (2 Nos.)

Rated capacity		=	1250 kVA
No.of transformers		=	2 Nos
Phase		=	3
Frequency		=	50
Cooling		=	ON
Volt at No Load	HV	=	6600
	LV	=	433
Full load Amperes	HV	=	109.3
	LV	=	1666
Rated temp. rise in oil		=	45°C
Rated temp. rise by resistance		=	55°C
Vector group ref No.		=	DY11
Qty of oil		=	925 l

PERFORMANCE DETAILS

1600 kVA Transformers

% Regulation at full load at 75°C at UPF/0.8 PF	=	1.35 at UPF. 4.57 at 0.8 PF
Maximum efficiency	=	99.67%
Load at which maximum efficiency occurs	=	38.4% of F.L

Appendix - 3/4 contd..

Efficiency at :

Full load - 0.8 PF/	= 98.32/98.66
75% load - UPF.	= 98.90/99.16
50% load - %	= 99.68/99.51

No load loss at rated voltage	= 2.8 kW
Full load loss at 75°C	= 19.0 kW

1250 kVA Transformers

% Regulation at full load at 75°C at UPF/ 0.8 pf	= 1.27 at UPF. 4.35 at 0.8 PF
Maximum efficiency	= 99.6%
Load at which maximum efficiency occurs	= 42.3% of F.L

Efficiency at :

Full load - 0.8 PF/	= 98.29/98.66
75% load - UPF.	= 98.94/99.15
50% load - %	= 99.38/99.50

No load loss at rated voltage	= 2.6 kW
Full load loss at 75°C	= 14.5 kW

1000 kVA Transformers

% Regulation at full load at 75°C at UPF/0.8 PF	= 2.307 at UPF. 3.819 at 0.8 PF
Maximum efficiency	= 99.46%
Load at which maximum efficiency occurs	= 48.2% of F.L

Efficiency at :

Full load - 0.8 PF/	= 98.28/98.62
75% load - UPF.	= 98.90/99.13
50% load - %	= 99.00/99.21

No load loss at rated voltage	= 2.6 kW
Full load loss at 75°C	= 11.20 kW

Appendix - 3/4 contd..

NAME PLATE DETAILS OF RUSSIAN FURNACE TRANSFORMER

A. 6.6 kV/11 kV System

Make		=	Bharat Bijlee
Capacity		=	1000 kVA
Primary/Secondary	Volt	=	6.6 /11 kV
	Amp	=	87.8/52.5
Impedance		=	5.13 %
Off load tap		=	+ 5% to - 5% (1 - 5)
Existing tap position		=	3
Core & Winding		=	1800 kg
Oil		=	750 kg
Total weight		=	4050 kg
Oil in lit.		=	875
Year of Manufacture		=	1977

B. 11/0.433 kV System

Make	=	USSR	
Capacity	=	1000 kVA	
Connection	=	Δ λ -11	
Impedance	=	7.06 %	
Active part	=	4154 kg	
Oil mass	=	2885 kg	
Total mass	=	9095	
Type	=	TMH3-1000/35-73T2	
Off load tap changer		Voltage	Current
Secondary			
Tap 1	=	517 V	1144 A
Tap 7	=	439 V	1215 A
Tap 14	=	382 V	1376 A
Primary	=	11,000 V	37 A

APPENDIX - 3/5

6.6 KV CAPACITORS - 4 BANKS

A. Bank :

Rated output	=	2016 kVAr
Rated voltage	=	7300 V
Rated current	=	155 A
No.of capacitor units/bank (distributed on 3 phases)	=	18
No.of capacitor units in series/ph	=	1
No.of capacitor units parallel for series group	=	6
Type of connection	=	λ

B. Capacitor Unit

Manufacturer's Name	=	Manohar bros.Pvt Ltd
Rated O/P	=	112 kVA
Rated Volt	=	4200
Rated Amp	=	26
Frequency	=	50 Hz
Di-electric	=	Mixed dielectric of polypropylene with interspaced neoter impregnated paper
Losses (Watts/kVAr)	=	0.6 W/kVAr

C. Series reactor	= 4 Nos.
Manufacturer's name	= P S Industrials
Rating	= 112/115 kVAr
Type	= Air cored oil cooled magnetically shielded
Volt/amp	= 7.3 kV - 154 Amps
Reactance	= 1.45 ohms

D. Residual voltage transformer - 4 Nos.

Manufacturer's name	= Gyro laboratories
Type	= Outdoor, oil cooled 6.6 kV class, 50 V/ph 5 limb (star-star -open delta)

RECORD OF POWER FAILURE/INTERRUPTION FROM APSEB
(JAN 1994-DEC 1994)

Month	Power failure Hrs	Power restrictions Hrs	Voltage dip No.of times
Jan 94	0.25	125	4
Feb *	0	45	-
Mar	1 min	384	-
Apr	2	96	-
May	-	45	2
Jun	-	-	-
Jul	11	-	-
Aug	-	-	-
Sep	-	40	-
Oct	8	-	-
Nov	-	15	-
Dec	-	-	-
Total	21.75	1050	-

* 40% power cut imposed from February 1994.

Appendix - 3/7 contd.

Time	132 kV System					Rectifier-1		Rectifier-2		DG Panel			6.6 kV Panel			Total plant load MW	
	kV	I	MW	MVA	PF	MW	A	MW	A	MW	MW	MW	PF	MW	I	APSEB DG	Rect. 1, 2 & Plat loads
20.00	128	40	9	9.7	0.99	6.0	115	5.5	100	3.7	3.4	3.5	0.92	5.5	110	19.6	17.4
21.00	128	40	9	7.7	0.99	6.5	115	5.0	100	3.7	3.4	3.5	0.92	5.0	100	19.6	16.5
22.00	128	40	9	9.7	0.98	6.5	115	5.0	100	3.7	3.4	3.5	0.92	5.0	100	19.5	16.5
23.00	129	40	9	10.7	0.98	6.5	115	5.2	100	3.7	3.4	3.5	0.92	5.3	100	19.6	17.1
24.00	128	40	10	10.2	0.98	7.1	130	5.8	115	3.7	3.4	3.5	0.92	5.8	100	20.5	18.7
01.00	128	40	10	10.3	0.98	7.1	130	5.8	115	3.7	3.4	3.5	0.92	5.8	115	20.5	18.7
02.00	128	40	10	10.6	0.98	7.1	130	5.8	115	3.7	3.4	3.5	0.92	5.8	115	20.6	18.7
03.00	129	40	10	10.3	0.98	7.1	130	5.8	115	3.7	3.4	3.5	0.92	5.8	115	20.5	18.7
04.00	128	40	10	10.3	0.98	7.1	130	5.8	115	3.7	3.4	3.5	0.92	5.8	115	20.6	18.7
05.00	129	40	10	10.3	0.98	7.0	130	5.8	115	3.7	3.4	3.5	0.92	5.9	115	20.5	18.7

APPENDIX - 3/8

SYSTEM PARAMETERS

132/33/6.6 kV - System

Details of Calculations

1. Load factor = $\frac{\text{Average load}}{\text{Peak load}}$

2. Loss load factor = $\frac{\Sigma\ I1^2 + I_2^2 + \ldots In^2}{Imax^2 \times 8760}$

3. Data for 1994-95

    a. Annual electricity consm. from APSEB = 1193.7 L.kWh

    b. Annual self generation = 227.18 L.kWh

    c. Maximum demand (highest) = 21840 kVA

    d. Average power factor = 0.95 & above

4. 132 kV system

    a. Annual load factor = 65.6%

    b. Annual loss load factor = 0.440 (Dec 95- 0.667)

5. 33/6.6 kV and 6.6 kV/433 V system

    a. Annual load factor = 86.99%

    b. Annual loss load factor = 0.756

6. Average cost of electricity (including demand and surcharges at revised tariff) = Rs.2.54 ps.per kWh

TRANSFORMER LOAD MANAGEMENT

DETAILS OF CALCULATIONS AND RESULTS OF COMPUTER RUN

1. Total losses P_{lt} of transformer during operation are :

   $P_{lt} = P_0 + (P_s/P_n)^2 \times P_{sc}$

2. Optimum loading ratio OLR = $\sqrt{P_0/P_{sc}}$

3. Load loss = $(P_s/P_n)^2 \times P_{sc} \times LLF \times UF$

Where,

P_{lt} = Total power loss in kW

P_0 = Open Circuit losses in kW

P_{sc} = Short circuit losses in kW

P_s = Actual load of transformer in kVA

P_n = Rated power of transformer in kVA

LLF = Loss Load Factor

UF = Utilisation Factor

A. 35/40 MVA transformer - 2 Nos.

Data

Considering the two transformers as one system :

F/L rating of transformer	= 35 MVA (ONAF) each
Capacity of transformation available	= 70 MVA
No load loss	= 36 kW

Appendix - 3/9 contd..

F/L load loss	= 326.74 kW
Optimum loading ratio	= 33.2%

Derived Data

Existing loading	= 19.11 MVA (27.3%)
Additional loads éxpected (Upgradation of electrolysis plant)	= 3 MVA
Proposed load	= 31.5%
Optimum loading ratio	= 33.2%
No load loss/annum	= 315360 kWA
Load loss/annum	= 124962 kWh
Total transformation loss/yr	= 440322 kWh

The transformer is optimally loaded.

APPENDIX - 3/10

LOADING OF 2 x 10/12.5 MVA 33/6.6 KV TRANSFORMERS FOR PLANT LOADS

Data

Considering two transformers as one system :

Transformation capacity = 20 MVA (ONAN)

Max. & Min load on plant = 6.12 MW (max), 4.2 MW (min)

Avg. load on plant = 5.2 MW

No load loss = 9.6 + 9.6
= 19.2 kW

F/l load loss = 55.08 + 55.08
= 110.16 kW

Optimum loading ratio = 41.7%

Analysis

Existing load = 26%

N/L loss/annum = 168192 kWh

Load loss/annum = 49317 kWh

Total transformation losses = 217509 kWh

Proposal

Loading one transformer and switching off the other transformer (in cyclic rotation of one week).

Appendix - 3/10 contd..

Load on one transformer	= 52%
N/L loss/annum	= 84096 kWh
Load loss/annum	= 98634 kWh
Total transformation losses	= 182730 kWh
Annual savings	= 34770 kWh
Savings due to implementation of proposal during non-monsoon months (8 months in a year)	= 25853 kWh

The loading of one transformer is practiced at present and the proposal may be continued for minimising transformation losses.

APPENDIX - 3/11

6.6 kV/ 433 VOLTS SYSTEM

LOADING PATTERN OF DISTRIBUTION TRANSFORMERS

Sl No.	Ref No.	Feeder details	Rating (kVA)	% loading	Calculated kVA load
1	X-30	Lead plant	1600	18-20	180
2	X-31	Zinc oxide	1600	5-10	160
3	X-41	Zinc oxide	1600	30	480
4	X-33	Lead smelter	1600	40	640
5	X-43	Lead smelter	1600	7-10	160
6	X-35	Roaster plant	1600	42.50	800
7	X-45	Roaster plant	1600	- *	-
8	X-36	Acid/cooling tower	1600	18-24	384
9	X-46	Acid/cooling tower	1600	- *	-
10	X-34	Cadmium plant	1000	10-20	200
11	X-44	Workshop	1000	26-29	290
12	X-32	Leaching plant	1600	28-35	560
13	X-42	Leaching plant	1600	17-35	560
14	X-37	Electrolysis	1600	11-18	288
15	X-47	Electrolysis	1250	44-46	575
16	X-48	Russian furnace	1000	53	530
17	X-39	Compressor	1250	5	62
18	X-49	Compressor	1250	28-32	400

* Transformers are idle charged.

Total transformation capacity available	**= 25950 kVA**
Max.load on plant transformers	= 6269 kVA
Max. load at 80% load factor (This load is apart from HT motor loads on 6.6 kV bus)	= 5015 kVA
Total load of HT motors on 6.6. kV bus	= 1692 kW (Refer Appendix - 4/1)
Load at 80% load factor	= 1353 kW ie., 1503 kVA @ 0.9 PF
Total plant load on 6.6 kV bus	= 6518 kVA

APPENDIX - 3/11

6.6 kV/ 433 VOLTS SYSTEM

LOADING PATTERN OF DISTRIBUTION TRANSFORMERS

Sl No.	Ref No.	Feeder details	Rating (kVA)	% loading	Calculated kVA load
1	X-30	Lead plant	1600	18-20	180
2	X-31	Zinc oxide	1600	5-10	160
3	X-41	Zinc oxide	1600	30	480
4	X-33	Lead smelter	1600	40	640
5	X-43	Lead smelter	1600	7-10	160
6	X-35	Roaster plant	1600	42.50	800
7	X-45	Roaster plant	1600	- *	-
8	X-36	Acid/cooling tower	1600	18-24	384
9	X-46	Acid/cooling tower	1600	- *	-
10	X-34	Cadmium plant	1000	10-20	200
11	X-44	Workshop	1000	26-29	290
12	X-32	Leaching plant	1600	28-35	560
13	X-42	Leaching plant	1600	17-35	560
14	X-37	Electrolysis	1600	11-18	288
15	X-47	Electrolysis	1250	44-46	575
16	X-48	Russian furnace	1000	53	530
17	X-39	Compressor	1250	5	62
18	X-49	Compressor	1250	28-32	400

: Transformers are idle charged.

Total transformation capacity available = 25950 kVA

Max.load on plant transformers = 6269 kVA

Max load at 80% load factor (This load is apart from HT motor loads on 6.6 kV bus) = 5015 kVA

Total load of HT motors on 6.6. kV bus = 1692 kW (Refer Appendix - 4/1)

Load at 80% load factor = 1353 kW ie., 1503 kVA @ 0.9 PF

Total plant load on 6.6 kV bus = 6518 kVA

Appendix - 3/11 contd..

6.6 kV/433 V DISTRIBUTION TRANSFORMER LOADING PARAMETERS

Time	ROASTER PLANT				LEAD PLANT						
	Date : 30-07-95				Date : 01-08-95						
	X - 35		X - 36		X - 33		X - 43		X - 30		
	Amp	Volt	Amp	Volt	Amp	Volt	Amp	Volt	Amp	Volt	PF
10.00	1100	440	480	420	780	430	250	440	400	440	0.88
11.00	1100	440	480	420	780	430	240	440	300	440	0.80
12.00	1100	440	480	420	780	430	240	440	300	440	0.81
13.00	1000	450	450	430	770	430	240	440	300	440	0.81
13.30	800	450	450	430	-	-	-	-	-	-	-
14.00	800	450	450	430	770	430	240	440	280	440	0.81
15.00	900	450	450	430	800	430	240	440	250	440	0.90
16.00	1000	450	450	430	800	440	240	440	250	440	0 88
17.00	1000	450	450	430	800	440	240	440	250	440	0.88
18.00	1050	440	500	420	800	440	240	440	250	440	0.88
19.00	1050	440	500	420	840	430	250	430	250	440	0.88
20.00	1050	440	500	420	800	440	250	430	250	440	0.88
21.00	1050	440	500	420	800	440	250	430	230	430	0.90
22.00	1000	440	500	430	800	440	250	430	230	430	0.90
23.00	1000	440	500	430	-	-	-	-	-	-	-
24.00	1000	450	500	430	-	-	-	-	-	-	

Time	LEACHING										ELECTROLYSIS			
	Compressor House Substation				Leaching PCC				Cadmium		Electrolysis-1		Electrolysis-2	
	X - 39		X - 49		X - 32		X - 42		X - 34		X - 37		X - 47	
	Amp	Volt	Amp	Volt	Amp	Volt	Amp	Volt	Amp	Volt	Amp	Volt	Amp	Volt
6.00	Lighting Loads 50 A	420	200	420	600	420	300	420	160	420	400	420	500	420
7.00		420	360	420	600	420	300	420	160	420	400	420	500	420
8.00		420	440	420	600	420	600	420	160	420	350	420	850	420
9.00		420	440	420	500	420	600	420	160	420	350	420	850	420
10.00		420	480	415	550	420	525	420	160	420	250	410	900	410
11.00		420	360	420	400	420	450	420	64	420	350	410	950	410
12.00		420	400	420	450	420	450	420	64	420	350	420	950	420
13.00		420	400	420	450	420	450	420	96	420	350	420	950	420
14.00		420	400	420	450	420	450	420	96	420	350	420	500	420

Appendix - 3/11 contd..

Time	WORKSHOP		ZINC OXIDE	
	X - 44		X-31	X-41$
	Amp	Volt	Amp*	Volt
09.00	310-360	440	500 - 700 (max.)	
11.00	350	435		
13.00	300	440		
14.00	325-350	442		
15.00	300	434		
16.00	330	436		
19.00	300-325	435		
20.00	280-300	424		
21.00	250-300	424		

* Idle charged
$ Data from past records since plant was under shutdown.

CALCULATION OF LOSSES WITH EXISTING 13 NOS. OF TRANSFORMERS AT LEAD, ZINC OXIDE, & UTILITY S/S

Transformer make = GEC

N/L loss/Tr. = 2.8 kW

Load loss/Tr. = 19.0 kW

Optimum loading ratio = 38.4%

Sl No	Data	Unit	Lead plant 3 Tr	Zinc Oxide 2 Tr.	Roaster/ Acid & CT
1	N/L loss	kW	8.4	5 6	11.2
2	Load loss	kW	57	38	57
3	Annual N/L loss	kWh	73584	49056	98112
4	Annual load loss	kWh	15735	10066	12919
	Total losses	kWh	89319	59122	111031

Proposal

a. To switch off one transformer (1600 kVA rating) each from Lead plant, utility (cooling tower and Zinc oxide) substations. The existing plant loads may be shared with the available transformers.

b. To switch off primary and secondary of above three plant transformers (1x1600 kVA rating) during non-monsoon months and take load on one transformer only.

Savings in no load losses (For 3 transformers switched off for 6 months) = 36288 kWh

Cost of savings per annum = Rs.137895/-

Cost of implementation = Nil

AJAX FURNACE

Phase	Amp	Volt	PF	kVA	kW
MELTING CYCLE - MAIN INCOMING POWER SUPPLY PARAMETERS					
R	643	484	0.95 L	96.6	96.1
Y	406	413	0.97 L	156.0	142.5
B	506	420	0.78 L	122.8	95.0
Total				375.4	333.1
POWER PARAMETERS- MEASUREMENTS MADE AFTER FURNACE TRANSFORMER					
R	498	525	0.96 L	96.8	96.1
Y	316	532	0.95 L	149.5	142.0
B	396	542	0.75 L	123.7	95.0
Total				370	333.1
HOLDING CYCLE - MAIN POWER SUPPLY PARAMETERS					
Phase	Amp	Volt	PF	kVA	kW
R	56.5	400	0.8 L	13.13	11.28
Y	55	395	0.8 L	12.87	11.78
B	56	399	0.8 L	13.36	10.79
Total				39.36	32.85

RUSSIAN FURNACE : Power Supply - Load Readings

Phase Inductor/Capacitor Currents	
Reactor	258 Amp
Capacitor - 1	183 Amp
Capacitor - 2	168 Amp

Appendix - 3/11 contd..

PF Compensating Capacitor Currents	
IC1 - 181	IC4 - 174.3
IC2 - 176	IC5 - 121.31
IC3 - 168.3	IC6 - 120.7

* IC1 to IC4 = 210 kVAr, 770 V,
** IC5 to C6 = 146 kVAr, 770 V

NAME PLATE DETAILS OF AUTO RECTIFIER TRANSFORMER

(Mfd by NGEF)

Auto-transformer		
Rated power	kVA	9460
Rated voltage HV	kV	33
Supply voltage variation	%	+10% - 15%
Rated voltage LV	kV	30.135 to 22.017
Connections HV LV	-	Star-star
Vector group symbol	-	λ0
Type of cooling	-	OFW
Rectifier Transformer		
Rated power	kVA	9460
Rated voltage HV LV	kV V	30.135 to 22.017 514.5 to 375.9
Connections HV LV	kV V	Delta Star
Vector group symbol	-	Dy-11
Impedance at 75°C		8%
Weights of transformer	kg	19600
Weight of oil	kg	9800
Total weight of complete transformer	kg	36000

APPENDIX - 3/13

APSEB TRIVECTOR METER READINGS

From 14.00 hours to 16.00 hours on 16-08-95

Time	kWh		kVAh		kVArh	
	Reading	Recorded	Reading	Recorded	Reading	Recorded
2.00	597398	-	617666	-	140679	-
2.15	397944	2880	617672	2880	140680	480
2.30	597950	2880	617679	3360	140682	960
2.45	597957	3360	617686	3360	140683	480
3.00	597964	3360	617693	3360	140684	480
3.15	597971	3360	617699	2880	140685	480
3.30	597977	2880	617706	3360	140686	480
3.45	597984	3360	617713	3360	140687	480
4.00	597990	2880	617719	2880	140688	480
Total	-	24960	-	25440	-	4320

Avg. PF for 2 hours = 0.981

No.of capacitor banks 'ON' = 2 banks

= 3112 kVAr(after derating)

Hourly PF recorded = 0.963

Load on APSEB = 12.5 MW

Load from DG Sets = 6.6 MW

Make of trivector meter = Duke Arnics, Hyderabad.

APPENDIX - 3/14

OBSERVATIONS OF PANEL READINGS (WITH AND WITHOUT 6.6 kV CAPACITOR BANKS

Both 132 kV/33 kV and 33/6.6 kV transformers were put on manual OLTC control, ie., at tap 14 and tap 5 respectively.

132/33 kV System

System	MW	MVAr	PF	V	Amps	Remarks
4 banks 'on'	12.0	2.0	0.99	131.0	56	
3 banks 'on'	13.2	4.0	0.95	132.0	60	
2 banks 'on'	13.0	4.0	0.97	132.0	62	
1 banks 'on'	13.0	5.8	0.95	131.0	64	
All banks 'off'	12.0	6.5	0.93	131.0	62	
4 banks 'on'	12.8	2.5	0.99	33.5	200	Instantaneous readings
3 banks 'on'	-	-	-	-	-	
2 banks 'on'	13.8	2.5	0.93	33.0	190	
1 banks 'on'	13.0	2.5	0.90	33.2	200	
All banks 'off'	12.8	2.5	0.86	33.0	210	

Appendix - 3/14 contd..

REACTIVE POWER MANAGEMENT

Load on APSEB	= 13.0 - 13.2 MW
Load supplied by DG sets	= 6.4 MW
Reactive power supplied from DG sets	= 2.73 MVA
Total load on plant	= 19.6 MW
PF of incomer	= 0.97 lag with (2 capacitor banks 'on')
Effective kVAr output capacitor banks at 6.6 kV	= 3112 kVAr(after derating)
Instantaneous load on 6.6 kV bus	= 4.7 MW
PF on 6.6 kV bus	= 0.96 lag

The reactive compensation on 6.6 kV bus is optimum (when 2 banks are switched 'on') and when 2 DG sets are operated at PF of 0.92 lag. The other two banks may be kept as standby. Additional reactive compensation is proposed at motor control centres to an extent of 550 kVAr (Section 4.0 and Appendix - 4/5).

Two banks of 200 kVAr each are available in Acid plant LT room, which are not connected to the system. The same may be reconnected in groups of 50/100 kVAr each at various MCC panels as recommended.

APPENDIX - 3/15

MEASUREMENTS TAKEN FROM KW/COSΦ DIGITAL INSTRUMENT WITH AND WITHOUT CAPACITOR BANKS ON 6.6 KV BUS

Capacitor bank status	Total kVAr	Effective kVAr at 6.6 kV	ENERCON METER 33 kV SYSTEM				MULTIMETER READINGS 6.6 kV SYSTEM	
			33 kV system		6.6 kV system		Measured	
			MW	PF	MW	PF	V	I
All banks 'on'	7840	6408	4.7	0.94 lead	4.94	0.94 lead	6.78	463
Bank No.1, 2, 3 'on'	5824	4760	4.7	0.99 lead	4.80	0.99 lead	6.66	435
Bank Nos.1 & 2 'on'	3808	3112	4.7	0.96 lag	4.763	0.94 lag	6.57	443
Bank 1 'on'	1904	1556	4.7	0.86	4.687	0.87	6.42	499
All banks 'off'	-	-	4.7	0.78	4.575	0.77	6.312	565

* Panel readings
** Measured across CT/PT of each cubicle.

Rated capacity of bank 1 & 2 = 1904 kVAr each

Rated capacity of bank 3 & 4 = 2016 kVAr each

APPENDIX - 3/16

TRANSFORMER OFF-LOAD TAP SETTINGS

Sl No.	Ref.	Feeder details	Rating (kVA)	Tap position	Voltages observed
1	X-30	Lead plant	1600	3	430-440
2	X-31	Zinc oxide	1600	3	430-440
3	X-41	Zinc oxide	1600	3	430-440
4	X-33	Lead smelter	1600	3	430-440
5	X-43	Lead smelter	1600	3	430-440
6	X-35	Roaster plant	1600	1	440-450
7	X-45	Roaster plant	1600	3	440-450
8	X-36	Acid/cooling tower	1600	2	420-430
9	X-46	Acid/cooling tower	1600	1	420-430
10	X-34	Cadmium plant	1000	2	410
11	X-44	Workshop	1000	3	424
12	X-32	Leaching plant	1600	2	420
13	X-42	Leaching plant	1600	2	420
14	X-37	Electrolysis	1600	1	410-420
15	X-47	Electrolysis	1250	1	410-420
16	X-48	Russian trfr.	1000	3	-
17	X-39	Compressor	1250	2	420
18	X-49	Compressor	1250	2	420

APPENDIX - 3/17

OBSERVATIONS OF VOLTAGE LEVELS MADE ON ET PLANT LOAD FEEDERS FOR IDENTIFYING NECESSITY OF EXCLUSIVE DISTRIBUTION TRANSFORMER FOR LOADS

Date : 01-08-95
Time : 09-50 AM

Data :

Details	Transformer Ref	Load (Amp)	Volt	Tap position
Leaching S/S	X-32	600	420	2
	X-42	500	420	2
Compressor S/S	X-39	off	-	2
	X-49	550	430	2

Terminal Voltage Measurements

Area of measurement	Voltage readings		
	Ph 1-2	Ph 2-3	Ph 3-1
Voltage at S/S motor starter	408-414 V	409-417 V	409-417 V
Bus bar voltage	410-414 V	412-416 V	411-415 V
E T Plant MCC (3 x 240 sqmm 3 runs are laid from leaching/compressor S/S)*			
150 HP motor-1 location			
DOL start condition	397 V lowest dip		
Loaded condition	403-407	403-409	402-408
150 HP motor -2 location			
DOL start condition	365 V lowest dip		
Loaded condition	399-401	404	401

* Total load on MCC = 400 Amps (max.)
** All motors are having DOL starting

Conclusion : The voltage levels for the ET plant are optimal. As such there is no necessity of providing transformer for these loads. However, if there is any separate load growth far from this, one 1600 kVA transformer may be dedicated for the loads.

LIST OF HT CABLES AND DISTRIBUTION LOSSES

From	To	Trfr/ Motor Ref.	Size of cable mm^2*	Length mts	Annual Losses kWh/yr
A. 33 kV Cable Losses					
HT panel	Trfr.	X-11	400	30	-
HT panel	Trfr.	X-21	400	30	917
Rect. cubicle	Trfr.	X-12	300	150	6854
Rect. cubicle	Trfr.	X-22	300	150	6854
10 MVA trfr	6.6 kV panel	-	400*	35	-
10 MVA trfr	6.6 kV panel	-	400*	35	847
B. 6.6 kV Transformer Cable Losses					
MRS	Trfr.	X-30	120	805	-
MRS	Trfr.	X-34	120	500	840
MRS	Trfr.	X-39	120	405	-
MRS	Trfr.	X-35	120	450	-
MRS	Trfr.	X-37	240	515	4326
MRS	Trfr.	X-36	240	155	260
MRS	Trfr.	X-33	240	580	1949
MRS	Trfr.	X-32	240	805	6762
MRS	Trfr.	X-31	240	450	3780
MRS	Trfr.	X-41	240	620	-
MRS	Trfr.	X-42	240	620	1042
MRS	Trfr.	X-43	240	450	3780
MRS	Trfr.	X-47	240	815	2738
MRS	Trfr.	X-48	240	165	1386
MRS	Trfr.	X-48	120	250	3780

Appendix - 3/18 contd..

From	To	Trfr/ Motor Ref.	Size of cable mm^2*	Length mts	Annual Losses kWh/yr
MRS	Trfr.	X-44	120	1000	1680
MRS	Trfr.	X-45	240	565	-
MRS	Trfr.	X-46	240	580	-
MRS	Trfr.	X-49	185	450	2268
MRS	Trfr.	% MRS	240	50	84
C. 6.6 KV MOTORS					
MRS	6.6 kV motor	Baghouse blower	150	1000	6720
MRS	6.6 kV motor	DD blower	150	1000	6720
MRS	6.6 kV motor	RC gas fan	150	900	6048
MRS	6.6 kV motor	Ball mill	120	450	3024
MRS	6.6 kV motor	RÄB	120	535	3595
MRS	6.6 kV motor	200 TPD SO_2 blower	120	665	4469
MRS	6.6 kV motor	50 TPD SO_2 blower	240	715	4805
MRS	6.6 kV motor	Compressor	150	150	1008
Total kWh losses in HT system					86549

* 1 cable only
* 5 runs of cables

APPENDIX - 4/1

INSTANTANEOUS MOTOR READINGS

Sl No	Application	Rated kW	kW	kVA	PF	I	V	Hz	% Loading
	ROASTER PLANT								
1	Intermittent Blower	187	108.9	124.2	0.86	178.0	403.6	48.2	58.23
2	Cooling Air Fan	60	6.1	19.2	0.23	28.0	404.0	48.2	10.1
3	Feed Water Pump	75	43.29	61.11	0.80	85.0	407.0	48.2	56.4
4	Circulating Water Pump	55	38.91	47.10	0.76	71.4	407.0	48.2	7.7
5	Bucket Elevator	7.5	3.51	5.4	0.40	7.5	408.7	48.2	46.8
6	Rotary Discharge	3.7	0.60	1.71	0.35	2.2	408.7	48.2	16.2
7	Screw Conveyor	5.5	0.99	3.03	0.25	4.3	408.7	48.2	18 0
8	Inclined Chain Conveyor	2.2	1.80	3.72	0.40	4.8	408.7	48.2	81.8
9	Calcine Slurry Pump 42	37	22.59	30.24	0.70	41.9	412.2	48.2	61.1
10	Table Feeder	7.5	1.14	4.23	0.20	5.8	408.7	48.2	15.2
11	Slinger Feeder	7.5	1.32	4.53	0.40	6.7	408.7	48.2	17.6
12	Cooling Process Water Pump 3	110	50.4	61.5	0.81	84.6	421.0	48.2	45.8
13	C.T. Process Water Pump 1	110	85.2	104.7	0.81	142.7	417.4	48.2	77.5
14	C.T. Process Water Pump 4	110	65.4	88.18	0.71	120.6	419.0	48.2	59.5
15	Acidic Fan Motor	45	6.09	20.4	0.32	28.7	422.6	48.2	13.5
16	Non-acidic Fan	45	7.17	18.03	0.37	25.0	422.6	48.2	15.9
17	Calcine Cold Water Pump	30	21.15	26.31	0.77	36.2	422.6	48.2	70.5
18	Calcine Hot Water Pump	18.5	12.90	16.56	0.52	23.0	420.8	48.2	69.7
19	Calcine CT Fan	7.5	3.6	5.79	0.58	8.0	420.8	48.2	48.0
	MERCURY RECOVERY								
1	Blower Motor	45	36.24	49.05	0.80	69.0	410.5	48.2	80.5

Appendix - 4/1 contd..

Sl No	Application	Rated kW	kW	kVA	PF	I	V	Hz	% Loading
	ACID PLANT								
1	DT Pump (200 TPD)	55	52.5	63.0	0.82	91.6	389.7	47.9	95.5
2	AT Pump (200 TPD)	75	44.37	53.01	0.81	77.0	389.7	47.9	59.16
3	Splasher Motor	5.5	1.80	5.43	0.26	7.5	433.0	47.9	32.7
4	Agitator	2.2	1.50	2.52	0.59	3.2	433.0	47.9	68.2
5	Booster Fan (200 TPD)	75	7.10	25.98	0.27	34.9	433.0	47.9	9.5
	BLEND YARD								
1	Hammer Mill	55	39.39	50.58	0.78	66.5	405.3	48.2	71.6
2	Belt Conveyor 27	15	4.62	11.13	0.45	16.0	407.0	48.2	30 8
3	Belt Conveyor 24	11	3.36	6.75	0.56	13.0	40.70	48.2	30.5
4	Belt Conveyor 26	7.5	1.47	4.08	0.24	6.9	405.3	48.2	19.6
5	Vibrating Screen	11	2.55	5.85	0.35	3.4	407.0	48.2	23.2
	PUMP HOUSE								
1	Filter Water Pump	75	33.39	41.91	0.78	58.3	415.7	47.9	44.5
2	Filter Water Pump	37	36.81	48.66	0.72	65.7	415.7	48.1	98.16
3	Emergency Water Pump	110	52.5	76.50	0.66	107.8	412.2	48.1	47.7
4	Rectifier Water Pump	37	19.8	25.38	0.73	35.2	408.8	48.1	52.8
5	Clarified Water Pump	75	48.0	69.00	0.67	99.2	408.8	47.9	64.0
6	Clarified Water Pump	75	51.03	60.30	0.83	85.8	403.5	47.9	68.04
7	Rectifier Water Pump	37	14.10	21.42	0.60	30.0	410.5	47.9	37.6
	CELL HOUSE								
1	Cooler No. 1	22	16.29	19.95	0.77	26.6	408.76	48	74.05
2	Cooler No. 2	18.5	3.9	12.72	0.35	18.1	408.76	48	21.08
3	Electrolyte Pump 82	37	27.96	32.49	0.84	44	410.50	48	74.56
4	Electrolyte Pump 83	37	28.55	33.09	0.84	46.7	410.50	48	76.16
5	C T Fan 4	22	13.59	21.12	0.61	29.8	407.03	48	61.77
6	Electrolyte Pump 85	37	29 1	32.1	0.84	50.0	419.00	48	77.60
7	Electrolyte Pump 81	37	27.8	36.10	0.75	47.0	414.00	48	74.00

Appendix - 4/1 contd..

Sl No	Application	Rated kW	kW	kVA	PF	I	V	Hz	% Loading
8	C T Fan 5	22	7.95	12.9	0.59	17.4	407.03	48	36.14
9	Hammer Mill	22	0.72	3.45	0.2	4.9	407.03	48	3 27
10	Electrolyte Pump 72	37	24.78	28.92	0.83	40.9	410.50	48	66.08
11	Electrolyte Pump 73	37	33.9	39.33	0.82	56.3	405.30	48	90.40
12	Electrolyte Pump 71	37	26.37	30.93	0 8	43.4	405.30	48	70.32
13	Gypsum Cooler	22	9.96	15.12	0.62	22.5	412.23	48	45.27
14	Cold Soln Pump 66b	15	8.88	12.9	0.68	18.7	420.89	47.9	59.20
15	Electrolyte Pump 48	15	8.01	13.59	0.61	19.3	393.18	48.2	53.40
16	New Cooler	30	9.96	15.6	0.58	21.8	410.50	47.9	33.20
17	Hot Soln. Pump 65 B	11	12.75	16.56	0.72	22.9	419.16	47.9	115.91
18	Electrolyte Pump 47	37	3.63	5.73	0.5	8	410.50	47.9	9.68
19	Electrolyte Pump 78	37	35.94	39.6	0.86	53.1	429.55	48.2	95.84
20	Electrolyte Pump 76	37	32.79	39.45	0.81	53	427.82	48	87.44
21	Electrolyte Pump 75	37	30	35.1	0.8	47.7	427.82	47.9	80.00
22	Electrolyte Pump 48	37	27.81	35.28	0.75	47.2	427.82	47.9	74.16
	LEACHING								
1	Dorr overflow Pump 16	18.5	7.2	15.9	0.41	20.4	434.74	48.2	38.92
2	Dorr overflow Pump 01	18 5	4.2	13.05	0.36	17.2	434.74	48.2	22.70
3	Pachuca Discharge Pump 08	18.5	10.5	17.82	0.57	25.5	436.48	48.1	56.76
4	Pachuca Discharge Pump 09	15	9.9	11.7	0.65	19	436.48	48	66.00
5	Neutral Dorr Pump 14a	15	10.71	15.42	0.61	21.5	419.16	48.8	71.40
6	Purification Pump Istg 16A	18.5	9.6	13.8	0.6	20.6	419.16	48	51.89
7	Purification Pump IIIstg 19A	18.5	10.02	15.12	0.61	20.5	419.16	48	54.16
8	Purification Pump IIstg 13	18.5	9.6	16.41	0.54	22.7	419.16	48	51.89
9	Heat Exchanger inlet	18.5	9.6	14 1	0.61	19.5	417.42	48	51.89
10	Acidic Door overflow	18.5	9.24	15.48	0.57	21.3	419.16	47.9	49.95
11	ZNO2 Ball Mill 31 Pump	18.5	9.18	15.24	0.5	21	419.16	47.9	49.62
12	ZNO2 Ball Mill 32 Pump	18.5	6.93	13.62	0.45	18.2	420.89	47.9	37.46

Appendix - 4/1 contd..

Sl No	Application	Rated kW	kW	kVA	PF	I	V	Hz	% Loading
13	ZNO2 Ball Mill 23 Pump	18.5	5.82	13.83	0.14	17.7	420.89	47.9	31.46
14	Pachuca discharge Pump 07	15	5.85	17.25	0.42	25.5	420.89	47.9	39 00
15	Pachuca discharge Pump 07A	15	9.36	13.08	0.6	17.9	419.16	47.9	62.40
16	Slime leaching inlet Pump 12	15	7.77	14.55	0.52	21.3	419.16	48.1	51 80
17	N1 Agitator Pachuca	22	2.67	15.18	0.18	20.3	426.08	48	12.14
18	N2 Agitator Pachuca	22	5.94	19.74	0.27	26.2	429.55	48	27.00
19	N3 Agitator Pachuca	22	5.1	15.9	0.31	21.3	427.82	48	23.18
20	N4 Agitator Pachuca	22	2.31	15.48	0.16	20.5	429.55	48	10.50
21	N6 Agitator Pachuca	22	2.79	14.58	0.19	19.7	427.82	48	12.68
22	N7 Agitator Pachuca	22	2.67	14.94	0.18	20.2	429.55	47.9	12.14
23	Agitator 43	7.5	3.69	6.9	0.56	9.1	429.55	47	49.20
24	Agitator 44	7.5	3.9	7.44	0.7	10	427.82	47.9	52.00
25	Agitator 45	7.5	3.6	6.3	0.43	8.3	427.82	47.9	48 00
26	Agitator 46	7.5	3.78	6.96	0.59	9.5	429.55	48	50.40
27	Agitator 47	7.5	3.9	5.01	0.67	6.7	429.55	48	52.00
28	Agitator 48	7.5	2.55	3.27	0.55	7.6	427.82	48	34.00
29	Slime Pachuca 1	22	3.3	9.93	0.38	12.3	429.55	47.9	15.00
30	Slime Pachuca 2	22	6.9	16.71	0.44	22.6	429.55	47.9	31.36
31	New Pachuca Discharge Pump	11	3.3	10.5	0.1	13	429.55	47.9	30.00
32	Dorr Pit Pump	18.5	10.5	19.23	0.51	25.4	434.74	47.9	56.76
	SFD								
1	Vacuum Pump New	187	54.18	146.4	0.34	196.4	431.28	47.8	28.97
2	Neutralisation tailing Pump	15	4.74	12.27	0.36	17.1	412.23	47.9	31.60
3	Buffer Tank Pump 07	9.3	4.14	9.72	0.45	13.4	410.50	47.9	44.52
4	Float scavenger cell 10	11	5.22	6.87	0.64	9.2	415.69	48	47.45
5	Float scavenger cell 11	11	4.41	6.6	0.65	9	412.23	48	40.09
6	Float scavenger cell 9	11	3.39	6.51	0.56	9.3	413.96	48	30.82

Appendix - 4/1 contd..

Sl No	Application	Rated kW	kW	kVA	PF	I	V	Hz	% Loading
7	Float scavenger cell 12	11	3.6	6.9	0.6	9.1	412.23	48	32.73
8	D5 overflow pit Pump	11	7.05	14.46	0.52	20	410.50	47.9	64.09
9	Float Rough Cell 1	11	4.5	7.02	0.63	9.6	412.23	48	40.91
10	Float Rough Cell 5	11	4.38	6.12	0.64	8.5	408.76	48	39.82
11	Float Rough Cell 2	11	4.11	6.09	0.48	8.5	412.23	48	37.36
12	Float Rough Cell 3	11	3.27	6.36	0.45	9	407.03	48	29.73
13	Agitator Lime Slur. Pump 14	11	3.51	6.96	0.53	9.8	410.50	48	31.91
14	Agitator Lime Slur Pump15	11	5 1	5.94	0.93	8.3	424.35	48	46.36
15	Agitator Buffer Tank	18.5	9.6	10.17	0.9	14	424.35	48	51.89
16	Vacuum Pump 1	110	51	62.7	0.81	86.6	422.62	48	46.36
17	Dorr 4 overflow Pump	11	8.94	11.94	0.66	16.2	422.62	48	81.27
18	Rectifier Return Pump	11	3.75	13.83	0.2	18.3	407.03	48	34.09
19	Filtrate Pump	7.5	4.83	6 36	0.62	8.5	427.82	48	64.40
	CADMIUM PLANT								
1	Agitator L1	15	1.5	15.3	0.12	20.8	433.01	48	10.00
2	Exhaust Fan	7.5	7.2	13.8	0.45	18.5	422.62	48	96.00
3	Agitator L3	5.5	2.67	3.87	0.8	5.2	429.55	48	48.55
	CHARGE PREPARATION								
1	Drum Mixer I stage	15	4.11	10.89	0.45	14.4	426.08	47.9	27.40
2	Paddle Mixer	7.5	1.47	4.95	0.1	6.7	424.35	47.9	19.60
3	Hammer Mill	45	7.8	20.79	0.45	29.5	417.42	47.9	17.33
4	Drum Mixer II stage	18.5	3.63	13.56	0.16	19	417.42	48	19.62
5	Belt Conveyor 10	7.5	7.02	8.85	0.74	11	424.35	47.9	93.60
	D L PLANT								
1	Sinter Breaker	55	3.87	26.58	0.13	36.1	424.35	47.9	7.04
2	Fresh Air Fan	37	29.01	34.2	0.79	46.7	422.62	48	78.41
3	Ignition Air Fan	22	16.74	17.01	0.67	30.5	422.62	47.9	76.09
4	Combustion Air Fan	9.3	3.06	3.42	0.87	4.5	424.35	47.9	32.90

Appendix - 4/1 contd..

Sl No	Application	Rated kW	kW	kVA	PF	I	V	Hz	% Loading
	CRUSHER HOUSE								
1	Belt Conveyor 11	7.5	3.75	6 63	0.48	8.6	426.08	47.9	50.00
2	Belt Conveyor 15	7.5	4.8	12.3	0.33	16.1	436.48	48	64.00
3	Double Deck Screen	18.5	5.4	13.14	0.38	17.3	436.48	48	29.19
4	Roll Crusher 1	18.5	3.6	12.57	0.26	16.9	434.74	47.9	19.46
5	Roll Crusher 2	18.5	3.9	12.69	0.29	16.9	434.74	47.9	21.08
6	Drum Cooler	15	3.39	11.61	0.24	15.6	433.01	47.9	22.60
7	R C Pump 1	37	4.62	7.32	0 52	8.8	426.08	48	12.49
	GAS CLEANING PLANT								
1	Hot Water Sump Pump 2	22	8.58	12.18	0.59	16.5	424.35	47.9	39.00
2	Hot Water Sump Pump 1	22	9.21	17.37	0.46	23.3	431.28	47.9	41.86
3	Hot Water Dewatering Pump	18.5	9.33	14.7	0.58	20.9	405.30	48	50.43
4	R C Pump 16	18.5	6.06	11.1	0.48	15	422.62	48	32.76
5	Stripper Feed Pump 22A	18.5	5.7	11.16	0.43	15	426.08	47.9	30.81
	NEW BLAST FURNACE								
1	Granulation Pit Pump 1	37	17.61	25.95	0.65	35.5	426.08	47.9	47.59
2	Granulation Pit Pump 3	37	18.69	28.71	0.62	39.5	417.42	48.1	50.51
3	Scrubber Pump 3	18.5	3.75	14.76	0.27	21	422.62	48	20.27
4	Fumes Exhaust Blower	22	11.7	19.53	0.6	25	426 08	48	53.18
5	Main Skip Hoist	18.5	6.9	16.26	0.39	22.6	407.03	48	37.30
6	Coke Ship Motor	11	0.63	8.61	0.11	12.1	413.96	48	5.73
7	Roots Blower	110	52.62	76.8	0.71	107	415.69	48	47.84
8	Fumes Exhaust Blower	110	63.6	81.3	0.75	110 4	426.08	48	57.82
9	Steam Exhaust Blower	18.5	7.86	16.17	0.37	22.6	427.82	48	42.49
	COOLING TOWER								
1	Cooling Tower Pump 1	75	75.6	91.5	0.82	119	443.41	47 9	100.80
2	Return Water Pump 2	5.5	3	4.05	0.61	5	436.48	47.9	54.55
3	Cooling Tower Fan 1	7.5	3.36	3.78	0.94	5.1	438.21	48	44.80
4	Cooling Tower Pump 2	75	66.9	77.1	0.84	106.3	424.35	48	89.20
5	Cooling Tower Pump 3	37	18.45	28.29	0.61	38	431.28	48	49.86

Appendix - 4/1 contd..

Sl No	Application	Rated kW	kW	kVA	PF	I	V	Hz	% Loading
	LEAD REFINERY								
1	Agitator D	37	20.04	30.33	0.62	40.2	433.01	47.9	54.16
2	Kettle Blower 2	18.5	10.56	13.77	0.6	19.6	419.16	48	57.08
3	Kettle Blower 5	18.5	8.76	13.26	0.6	18.3	415.69	47.9	47.35
4	Kettle Blower 6	18 5	7.53	12.66	0.53	13	438.21	47.9	40.70
5	Kettle Blower 8	18.5	8.7	12.93	0.53	17.6	434.74	47.9	47.03
6	Kettle Blower 7	18.5	12	15.63	0.66	20.7	436.48	48	64.86
7	Agitator E	37	16.26	25.71	0.54	31	422.62	48.1	43.95
8	Vacuum Dezincing	22	2.1	8.97	0.16	12	422.62	48	9.55
	EFFLUENT TREATMENT PLANT								
1	Vacuum Pump 2	110	51.84	54.66	0.94	74	386.25	48	47.13
2	Horizontal Pump 3	11	5.94	9.03	0.6	12.9	405.30	48.1	54.00
3	Horizontal Pump 1	11	3.9	7.14	0.55	10.1	407.03	48.1	35.45
4	Air Blower 1	15	5.82	11.16	0.42	15.9	403.57	48	38.80
5	Lime Agitator Pump 2	5.5	2.07	3.33	0.44	4.8	405.30	48	37.64
6	Lime Agitator Pump 1	5.5	1.95	3.72	0.34	5.5	405.30	48	35.45
7	FD Pump 3	30	8.73	14.31	0.62	20.6	400.10	48	29.10
8	Storm Pump	15	7.02	9.3	0.7	13.3	408.76	48	46.80
9	Reaction Tank Istg Agitator	5.5	4.08	6.84	0.76	9.1	405.30	48	74.18
10	Reaction Tank IIstg Agitator	5.5	1.47	3.45	0.29	5.2	408.76	48	26.73
11	C2 underflow pump	5.5	2.04	3.12	0.39	4.2	408.76	48	37.09
	D G POWER HOUSE								
1	CT Fan 5	22	21.1	24.1	0.85	34.3	409	48	96.0
2	CT Fan 4	22	19.5	24.0	0.80	33.4	409	48	89.6

Appendix - 4/1 contd..

Sl No	Application	Rated kW	kW	kVA	PF	I	V	Hz	% Loading
	H T MOTORS								
1	Compressor	522	389	452	0.86	39	6600	50	75
2	Ball Mill	380	300	405	0.74	37	6600	48	79
3	RC Gas Fan	240	139.5	196	0.71	17.5	6600	48	58
		250	172.5	201	0.86	18.0	6600	48	69
4	Roaster Air Blower	250	147.0	175	0.84	16.0	6600	48	59
5	SO_2 Blower (200 TPD)	500	279	367	0.76	32	6600	48	56
6	SO2 Blower (50 TPD)	380	190.5	334	0.57	29	6600	48	50
7	Baghouse Blower	262	88	160	0.55	14	6600	48	34

APPENDIX - 4/2

ESTIMATED OPERATING EFFICIENCY OF MOTORS

Application	Rated kW	Rated Efficiency	Actual kW	Power on full load	Demand Factor	Estimated Efficiency
ROASTER PLANT						
Bucket Elevator	7.5	85	3.51	8.82	0.40	82
Rotary Discharge	3.7	83	0.6	4.46	0.13	51
Screw Conveyor	5.5	85	0.99	6.47	0.15	61
Inclined Chain conveyor	2.2	79	1.8	2.78	0.65	79
Table Feeder	7.5	85	1.14	8.82	0.13	55
Slinger Feeder	7.5	85	1.32	8.82	0.15	61
Calcine cold Water Pump	30	89	21.15	33.71	0.63	89
Calcine Hot Water Pump	18.5	89	12.9	20.79	0.62	89
Calcine CT Fan	7.5	85	3.6	8.82	0.41	82
ACID PLANT						
Splasher Motor	5.5	85	1.8	6.47	0.28	77
Agitator	2.2	79	1.5	2.78	0.54	78
BLEND YARD						
Belt Conveyor 27	15	88	4.62	17.05	0.27	81
Belt Conveyor 24	11	88	3.36	12.50	0.27	81
Belt Conveyor 26	7.5	85	1.47	8.82	0.17	64
Vibrating Screen	11	88	2.55	12.50	0.20	76
CELL HOUSE						
Cooler No. 1	22	89	16.29	24.72	0.66	89
Cooler No. 2	18.5	89	3.9	20.79	0.19	76
C T Fan 4	22	89	13.59	24.72	0.55	89
C T Fan 5	22	89	7.95	24.72	0.32	85
Hammer Mill	22	89	0.72	24.72	0.03	-44
Gypsum Cooler	22	89	9.96	24.72	0.40	87
Cold Soln pump 66b	15	88	8.88	17.05	0.52	87

Appendix - 4/2 contd..

Application	Rated kW	Rated Efficiency	Actual kW	Power on full load	Demand Factor	Estimated Efficiency
Electrolyte pump 48	15	88	8.01	17.05	0.47	87
New Cooler	30	89	9.96	33.71	0.30	84
Hot Soln. Pump 65 B	11	88	12.75	12.50	1.02	88
LEACHING						
Dorr overflow Pump16	18.5	89	7.2	20.79	0.35	86
Dorr overflow Pump01	18.5	89	4.2	20.79	0.20	78
Pachuca Discharge Pump08	18.5	89	10.5	20.79	0.51	88
Pachuca Discharge Pump09	15	88	9.9	17.05	0.58	88
Neutral Dorr Pump14a	15	88	10.71	17.05	0.63	88
Purification Pump Istg16A	18.5	89	9.6	20.79	0.46	88
Purification Pump IIstg19A	18.5	89	10.02	20.79	0.48	88
Purification Pump IIstg13	18.5	89	9.6	20.79	0.46	88
Heat Exchanger inlet	18.5	89	9.6	20.79	0.46	88
Acidic Door overflow	18.5	89	9.24	20.79	0.44	88
ZNO2 Ball Mill 31Pump	18.5	89	9.18	20.79	0.44	88
ZNO2 Ball Mill 32Pump	18.5	89	6.93	20.79	0.33	85
ZNO2 Ball Mill 23Pump	18.5	89	5.82	20.79	0.28	83
Pachuca discharge Pump 07	15	88	5.85	17.05	0.34	84
Pachuca discharge Pump 07A	15	88	9.36	17.05	0.55	88
Slime leaching inlet Pump12	15	88	7.77	17.05	0.46	87
N1 Agitator Pachuca	22	89	2.67	24.72	0.11	61
N2 Agitator Pachuca	22	89	5.94	24.72	0.24	81
N3 Agitator Pachuca	22	89	5.1	24.72	0.21	78
N4 Agitator Pachuca	22	89	2.31	24.72	0.09	55
N6 Agitator Pachuca	22	89	2.79	24.72	0.11	62
N7 Agitator Pachuca	22	89	2.67	24.72	0.11	61
Agitator 43	7.5	85	3.69	8.82	0.42	82
Agitator 44	7.5	85	3.9	8.82	0.44	83
Agitator 45	7.5	85	3.6	8 82	0.41	82

Appendix - 4/2 contd..

Application	Rated kW	Rated Efficiency	Actual kW	Power on full load	Demand Factor	Estimated Efficiency
Agitator 46	7.5	85	3.78	8.82	0.43	83
Agitator 47	7.5	85	3.9	8.82	0.44	83
Agitator 48	7.5	85	2.55	8.82	0.29	78
Slime Pachuca 1	22	89	3.3	24.72	0.13	68
Slime Pachuca 2	22	89	6.9	24.72	0.28	83
New Pachuca Discharge Pump	11	88	3.3	12.50	0.26	81
Dorr Pit Pump	18.5	89	10.5	20.79	0.51	88
SFD						
Neutralisation tailing Pump	15	88	4.74	17.05	0.28	82
Buffer Tank Pump 07	9.3	87	4.14	10.69	0.39	84
Float scavenger cell 10	11	88	5.22	12.50	0.42	86
Float scavenger cell 11	11	88	4.41	12.50	0.35	84
Float scavenger cell 9	11	88	3.39	12.50	0.27	81
Float scavenger cell 12	11	88	3.6	12.50	0.29	82
D5 overflow pit Pump	11	88	7.05	12.50	0.56	88
Float Rough Cell 1	11	88	4.5	12.50	0.36	85
Float Rougn Cell 5	11	88	4.38	12.50	0.35	84
Float Rough Cell 2	11	88	4.11	12.50	0.33	84
Float Rough Cell 3	11	88	3.27	12.50	0.26	81
Agitator Lime Slur. Pump 14	11	88	3.51	12.50	0.28	82
Agitator Lime Slur. Pump 15	11	88	5.1	12.50	0.41	86
Agitator Buffer Tank	18.5	89	9.6	20.79	0.46	88
Dorr 4 overflow Pump	11	88	8.94	12.50	0.72	88
Rectifier Return Pump	11	88	3.75	12.50	0.30	83
Filtrate Pump	7.5	85	4.83	8.82	0.55	84
CADMIUM PLANT						
Agitator L1	15	88	1.5	17.05	0.09	48
Exhaust Fan	7.5	85	7.2	8.82	0.82	85
Agitator L3	5.5	85	2.67	6.47	0.41	82

Appendix - 4/2 contd..

Application	Rated kW	Rated Efficiency	Actual kW	Power on full load	Demand Factor	Estimated Efficiency
CHARGE PREPARATION						
Drum Mixer I stage	15	88	4.11	17.05	0.24	79
Paddle Mixer	7.5	85	1.47	8.82	0.17	64
Drum Mixer II stage	18.5	89	3.63	20.79	0.17	75
Belt Conveyor 10	7.5	85	7.02	8.82	0.80	85
D L PLANT						
Ignition Air Fan	22	89	16.74	24.72	0.68	89
Combustion Air Fan	9.3	87	3.06	10.69	0.29	80
CRUSHER HOUSE						
Belt Conveyor 11	7.5	85	3.75	8.82	0.43	83
Belt Conveyor 15	7.5	85	4.8	8.82	0.54	84
Double Deck Screen	18.5	89	5.4	20.79	0.26	82
Roll Crusher 1	18.5	89	3.6	20.79	0.17	75
Roll Crusher 2	18.5	89	3.9	20.79	0.19	76
Drum Cooler	15	88	3.39	17.05	0.20	76
GAS CLEANING PLANT						
Hot H20 Sump Pump 2	22	89	8.58	24.72	0.35	86
Hot H20 Sump Pump 1	22	89	9.21	24.72	0.37	86
Hot H20 Dewatering Pump	18.5	89	9.33	20.79	0.45	88
R CPump 1B	18.5	89	6.06	20.79	0.29	84
striPumper Feed Pump 22A	18.5	89	5.7	20.79	0.27	83
NEW BLAST FURNACE						
Scrubber Pump 3	18.5	89	3.75	20.79	0.18	76
Fumes Exhaust Blower	22	89	11.7	24.72	0.47	88
Main Skip Hoist	18.5	89	6.9	20.79	0.33	85
Coke Ship Motor	11	88	0.63	12.50	0.05	9
Steam Exhaust Blower	18.5	89	7.86	20.79	0.38	86
COOLING TOWER						
Return Water Pump 2	5.5	85	3	6.47	0.46	83
Cooling Tower Fan 1	7.5	85	3.36	8.82	0.38	81

Appendix - 4/2 contd..

Application	Rated kW	Rated Efficiency	Actual kW	Power on full load	Demand Factor	Estimated Efficiency
LEAD REFINERY						
Kettle Blower 2	18.5	89	10.56	20.79	0.51	89
Kettle Blower 5	18.5	89	8.76	20.79	0.42	97
Kettle Blower 6	18.5	89	7.53	20.79	0.36	86
Kettle Blower 8	18.5	89	8.7	20.79	0.42	87
Kettle Blower 7	18.5	89	12	20.79	0.58	89
Vacuum Dezincing	22	89	2.1	24.72	0.08	58
ETP						
Horizontal Pump3	11	88	5.94	12.50	0.48	87
Horizontal Pump1	11	88	3.9	12.50	0 31	83
Air Blower 1	15	88	5.82	17.05	0.34	84
Lime Agitator Pump2	5.5	85	2.07	6.47	0.32	79
Lime Agitator Pump1	5.5	85	1.95	6.47	0.30	76
FD Pump3	30	89	8.73	33.71	0.26	82
Storm Pump	15	88	7.02	17.05	0.41	86
Reaction Tank Istg Agitator	5.5	85	4.08	6.47	0.63	85
Reaction Tank IIstg Agitator	5.5	85	1.47	6.47	0.23	73
C2 underflow Pump	5.5	85	2.04	6.47	0.32	79

APPENDIX - 4/3

REPLACEMENT OF EXISTING MOTORS WITH HIGH EFFICIENCY MOTORS

I. Hot Solution Pump 65B Motor

INPUT DATA			
Capacity of motor	:	H	11.00 kW
Efficiency of a standard motors	:	ES	0.88
Efficiency of a high eff motors	:	EH	0.90
Power factor of std motor	:	PS	0.86
Power factor of high eff. motor	:	PH	0.88
Energy rate (Rs per unit)	:	R	3.5
Maximum demand rate (Rs/kVA-month)	:	DS	0
Operating hours per year	:	N	7000.00
Cost of std motor in Rs.	:	CS	15000.00
Cost of high eff motor in Rs.	:	HS	18000.00
Annual capital charges (ratio)	:	F	0.15
ANALYSIS			
Incremental efficiency	:	X	0.023
Incremental cost	:	A	0.200
Gross savings			
Savings in energy (kWh/year)	:	S	1989
A) Savings in energy (Rs/year)	:	A1	6960
Savings in demand (kVA)	:	D	1.54
B) Cost savings in demand (Rs/year)	:	B	0
Total gross savings (Rs/year)	:	TS	6960
C) Incremental cost of high eff motor (Rs./year)	:	C	450.00
Net savings (Rs/year)	:	NS	6510
Return on investment		RI	
A) For new high eff motor	:		232.01
B) Replacement of existing motor with high eff motor	:		46.40
Simple payback period (years)	:		2.76

Appendix - 4/3 contd..

Specimen Calculations :

$$X = \frac{EH - ES}{ES} = \frac{0.9 - 0.88}{0.88} = 0.023$$

$$A = \frac{HS - CS}{CS} = \frac{18000 - 15000}{15000} = 0.2$$

$$S = \frac{H \times N \times X}{EH \times (1+X)} = \frac{11 \times 7000 \times 0.023}{0.9 \times (1 + 0.023)} = 1989 \text{ kWh/yr}$$

$$A1 = S \times R = 1923.23 \times 2.54 = Rs.6960$$

$$DS = \frac{H \times X \times (X+2)}{ES \times PS + (1+X)^2} = \frac{11 \times 0.023 \times (0.023+2)}{0.88 \times 0.86 + (1+0.023)^2} = 1.54 \text{ kVA}$$

$$B = DS \times 12 \times D = 0.64 \times 12 \times 110 = Rs.843.24$$

$$TS = A1+B = 6960 + 0 = Rs.6960$$

$$C = CS \times A \times F = 15000 \times 0.2 \times 0.15 = Rs.450$$

$$NS = A1 + B - C = 6960 - 450 = Rs.6510$$

RI for new high efficient motor :

$$\frac{NS + (CS \times A \times F)}{CS \times A} = \frac{5278.97 + (15000 \times 0.2 \times 0.15)}{15000 \times 0.2} \times 100 = 232.01$$

RI for replacement of existing motor with high efficient motor :

$$\frac{NS + (CS \times A \times F)}{(CS \times A) + (0.8 \times CS)} = \frac{6510 + (15000 \times 0.2 \times 0.15)}{(15000 \times 0.2) + (0.8 \times 15000)} \times 100 = 46.40$$

Payback period = 2.7 years

Appendix - 4/3 contd..

II. Splasher Motor

INPUT DATA			
Capacity of motor	:	H	5.50 kW
Efficiency of a standard motors	:	ES	0.85
Efficiency of a high eff motors	:	EH	0.87
Power factor of std motor	:	PS	0.82
Power factor of high eff. motor	:	PH	0.85
Energy rate (Rs per unit)	:	R	3.50
Maximum demand rate (Rs/kVA-month)	:	D	0
Operating hours per year	:	N	4000.00
Cost of std motor in Rs.	:	CS	6000.00
Cost of high eff motor in Rs.	:	HS	9000.00
Annual capital charges (ratio)	:	F	0.15
ANALYSIS			
Incremental efficiency	:	X	0.024
Incremental cost	:	A	0.500
Gross savings			
Savings in energy (kWh/year)	:	S	609
A) Savings in energy (Rs/year)	:	A1	2131
Savings in demand (kVA)	:	DS	1.30
B) Cost savings in demand (Rs/year)	:	B	0
Total gross savings (Rs/year)	:	TS	2131
C) Incremental cost of high eff motor (Rs./year)	:	C	450
Net savings (Rs/year) (A+B-C)	:	NS	1681
Return on investment		RI	
A) For new high eff motor	:		71.05
B) Replacement of existing motor with high eff motor	:		27.33
Simple payback period (years)	:		5.35

Appendix - 4/3 contd..

III. Neutral Dorr Overflow Pump Motor

INPUT DATA			
Capacity of motor	:	H	15.00 kW
Efficiency of a standard motors	:	ES	0.88
Efficiency of a high eff motors	:	EH	0.90
Power factor of std motor	:	PS	0.86
Power factor of high eff. motor	:	PH	0.88
Energy rate (Rs per unit)	:	R	3.5
Maximum demand rate (Rs/kVA-month)	:	D	0
Operating hours per year	:	N	7000.00
Cost of std motor in Rs.	:	CS	18000.00
Cost of high eff motor in Rs.	:	HS	22000.00
Annual capital charges (ratio)	:	F	0.15
ANALYSIS			
Incremental efficiency	:	X	0.023
Incremental cost	:	A	0.222
Gross savings			
Savings in energy (kWh/year)	:	S	2712
A) Savings in energy (Rs/year)	:	A1	9491
Savings in demand (kVA)	:	D	1.72
B) Cost savings in demand (Rs/year)	:	B	0
Total gross savings (Rs/year)	:	TS	9491
C) Incremental cost of high eff motor (Rs./year)	:	C	600
Net savings (Rs/year) (A+B-C)	:	NS	8891
Return on investment		RI	
A) For new high eff motor	:		237.28
B) Replacement of existing motor with high eff motor	:		51.58
Simple payback period (years)	:		2.47

IV. Calcine Hot Water Pump Motor

INPUT DATA			
Capacity of motor	:	H	18.50 kW
Efficiency of a standard motors	:	ES	0.89
Efficiency of a high eff motors	:	EH	0.91
Power factor of std motor	:	PS	0.88
Power factor of high eff. motor	:	PH	0.90
Energy rate (Rs per unit)	:	R	3.5
Maximum demand rate (Rs/kVA-month)	:	D	0
Operating hours per year	:	N	6000.00
Cost of std motor in Rs.	:	CS	23000.00
Cost of high eff motor in Rs.	:	HS	35000.00
Annual capital charges (ratio)	:	F	0.15
ANALYSIS			
Incremental efficiency	:	X	0.022
Incremental cost	:	A	0.522
Gross savings			
Savings in energy (kWh/year)	:	S	2803
A) Savings in energy (Rs/year)	:	A1	9809
Savings in demand (kVA)	:	DS	1.88
B) Cost savings in demand (Rs/year)	:	B	0
Total gross savings (Rs/year)	:	TS	9809
C) Incremental cost of high eff motor (Rs./year)	:	C	1800
Net savings (Rs/year) (A+B-C)	:	NS	8009
Return on investment		RI	
A) For new high eff motor	:		81.74
B) Replacement of existing motor with high eff motor	:		32.27
Simple payback period (years)	:		4.37

Appendix - 4/3 contd...

V. Filter Water Pump Motor

INPUT DATA			
Capacity of motor	:	H	50.00 kW
Efficiency of a standard motors	:	ES	0.90
Efficiency of a high eff motors	:	EH	0.92
Power factor of std motor	:	PS	0.88
Power factor of high eff motor	:	PH	0.90
Energy rate (Rs per unit)	:	R	3.5
Maximum demand rate (Rs/kVA-month)	:	D	0
Operating hours per year	:	N	8000.00
Cost of std motor in Rs.	:	CS	60000.00
Cost of high eff motor in Rs.	:	HS	75000.00
Annual capital charges (ratio)	:	F	0.15
ANALYSIS			
Incremental efficiency	:	X	0.022
Incremental cost	:	A	0.250
Gross savings			
Savings in energy (kWh/year)	:	S	9877
A) Savings in energy (Rs/year)	:	A1	34568
Savings in demand (kVA)	:	D	3.24
B) Cost savings in demand (Rs/year)	:	B	0
Total gross savings (Rs/year)	:	TS	34568
C) Incremental cost of high eff motor (Rs./year)	:	C	2250
Net savings (Rs/year) (A+B-C)	:	NS	32318
Return on investment		RI	
A) For new high eff motor	:		230.45
B) Replacement of existing motor with high eff motor	:		54.87
Simple payback period (years)	:		2.32

Appendix - 4/3 contd..

VI. Clarified Water Pump Motor

INPUT DATA			
Capacity of motor	:	H	75.00 kW
Efficiency of a standard motors	:	ES	0.91
Efficiency of a high eff motors	:	EH	0.93
Power factor of std motor	:	PS	0.88
Power factor of high eff. motor	:	PH	0.90
Energy rate (Rs per unit)	:	R	3.5
Maximum demand rate (Rs/kVA-month)	:	D	0
Operating hours per year	:	N	8000.00
Cost of std motor in Rs.	:	CS	110000.00
Cost of high eff motor in Rs.	:	HS	165000.00
Annual capital charges (ratio)	:	F	0.15
ANALYSIS			
Incremental efficiency	:	X	0.022
Incremental cost	:	A	0.500
Gross savings			
Savings in energy (kWh/year)	:	S	14491
A) Savings in energy (Rs/year)	:	A1	50719
Savings in demand (kVA)	:	D	4.27
B) Cost savings in demand (Rs/year)	:	B	0
Total gross savings (Rs/year)	:	TS	50719
C) Incremental cost of high eff motor (Rs./year)	:	C	8250
Net savings (Rs/year) (A+B-C)	:	NS	42469
Return on investment		RI	
A) For new high eff motor	:		92.22
B) Replacement of existing motor with high eff motor	:		35.47
Simple payback period (years)	:		4.5

Appendix - 4/3 contd..

VII. CT Process Water Pump Motor

INPUT DATA			
Capacity of motor	:	H	110.00 kW
Efficiency of a standard motors	:	ES	0.91
Efficiency of a high eff motors	:	EH	0.93
Power factor of std motor	:	PS	0.90
Power factor of high eff. motor	:	PH	0.92
Energy rate (Rs per unit)	:	R	3.5
Maximum demand rate (Rs/kVA-month)	:	D	0
Operating hours per year	:	N	7000.00
Cost of std motor in Rs.	:	CS	165000.00
Cost of high eff motor in Rs.	:	HS	200000.00
Annual capital charges (ratio)	:	F	0.15
ANALYSIS			
Incremental efficiency	:	X	0.022
Incremental cost	:	A	0.212
Gross savings			
Savings in energy (kWh/year)	:	S	18597
A) Savings in energy (Rs/year)	:	A1	65089
Savings in demand (kVA)	:	D	5250
B) Cost savings in demand (Rs/year)	:	B	0
Total gross savings (Rs/year)	:	TS	65089
C) Incremental cost of high eff motor (Rs./year)	:	C	6000
Net savings (Rs/year) (A+B-C)	:	NS	59089
Return on investment	:	RI	216.22
A) For new high eff motor	:		
B) Replacement of existing motor with high eff motor	:		51.22
Simple payback period (years)	:		3.10

Appendix - 4/3 contd..

VIII. Emergency Water Pump Motor

INPUT DATA			
Capacity of motor	:	H	110.00 kW
Efficiency of a standard motors	:	ES	0.91
Efficiency of a high eff motors	:	EH	0.93
Power factor of std motor	:	PS	0.90
Power factor of high eff. motor	:	PH	0.92
Energy rate (Rs per unit)	:	R	3.5
Maximum demand rate (Rs/kVA-month)	:	D	0
Operating hours per year	:	N	8000.00
Cost of std motor in Rs.	:	CS	165000.00
Cost of high eff motor in Rs.	:	HS	200000.00
Annual capital charges (ratio)	:	F	0.15
ANALYSIS			
Incremental efficiency	:	X	0.022
Incremental cost	:	A	0.212
Gross savings			
Savings in energy (kWh/year)	:	S	21253
A) Savings in energy (Rs/year)	:	A1	74387
Savings in demand (kVA)	:	D	5.88
B) Cost savings in demand (Rs/year)	:	B	0
Total gross savings (Rs/year)	:	TS	74387
C) Incremental cost of high eff motor (Rs./year)	:	C	5250
Net savings (Rs/year) (A+B-C)	:	NS	69137
Return on investment		RI	
A) For new high eff motor	:		212.53
B) Replacement of existing motor with high eff motor	:		44.54
Simple payback period (years)	:		2.89

APPENDIX - 4/4

MAIN INCOMER POWER READINGS

Sl No.	MCC NO	kW	kVA	PF	I	V	HZ
1	RP 1	42.84	97.8	0.74	164.4	406.55	48.2
2	Roaster	24.54	42.27	0.55	58.3	420.39	48.2
3	Acid plant	66.66	140.4	0.48	183	425.58	47.9
4	Rectifier room	166.8	231	0.69	332	410.01	48
5	Pump house	58.8	77.28	0.74	109	413.47	47.9
6	Cell house	51.3	73.8	0.64	96	411.74	47.9
7	Zinc casting	22.5	40.32	0.61	49.2	411.74	47.9
8	MCC 402	46.14	80.88	0.56	110.6	425.58	48.1
9	MCC 401	58.32	111.66	0.47	151.4	429.04	48.1
10	MCC 403	44.4	155.4	0.34	210	427.31	47.9
11	English Electric 2	78.6	120.9	0.61	170.1	410.01	48
12	English Electric 1	28.2	60.96	0.4	89.2	423.85	48
13	Powergear	108	448.2	0.63	145.5	427.804	48
14	New SFD	21.51	34.47	0.6	46.4	430.77	48
15	Coimbatore	19.8	39.33	0.46	52.2	432.5	48
16	Rotary furnace MCC	24.06	38.1	0.64	50.4	420.876	48
17	MCC 604	48	135.75	0.39	182.5	434.732	47.9
18	MCC 607	38.16	62.1	0.67	90	433	48
19	MCC 605	8.94	10.02	0.82	15.2	427.804	47.9
20	ETP MCC	113.64	175.2	0.62	240	408.752	48

APPENDIX - 4/5

POWER FACTOR IMPROVEMENT

A. Requirement of Capacitors

C_{kVAr} = kW ($\tan \phi_1 - \tan \phi_2$)

Where,
C_{kVAr} = Capacitor required

kW = Active power

ϕ_1 = Load pf angle (existing)

ϕ_2 = Desired pf angle

$$\text{Reduction in loss} = \left(1 - \frac{\cos \phi 1}{\cos \phi 2}\right)$$

MCC No	Measured				Capacity in kVAr	After installing Capacitor		Savings		Cost of implementation (Rs.)
	kW	kVA	I	PF		I	kVA	kVA	Cost (Rs.)	
RP 1	42.84	97.8	164.4	0.74	7	75.96	55.26	42.54	4680	1702
Roaster	24.54	42.27	58.3	0.55	19	42.08	30.61	11.66	1282	4715
Acid plant	66.66	140.4	183	0.48	72	112.91	82.14	58 26	6409	17959
Rectifier room	166.8	231	332	0 69	50	293.27	213.33	17.67	1943	12468
Pump house	58.8	77.28	109	0.74	9	102.52	74.57	2.71	298	2336
Cell house	51.3	73.8	96	0.64	23	89.82	65.34	8.46	931	5779
Zinc casting	22.5	40.32	49.2	0.61	12	39.39	28.66	11.66	1283	3098
MCC 402	46.14	80.88	110 6	0.56	34	78.15	56.85	24.03	2643	8414
MCC 401	58.32	111.66	151.4	0.47	66	97.99	71.28	40.38	4442	16446
MCC 403	44.4	155.4	210	0.34	90	74 90	54.49	100.91	11100	22377
English Electric 2	78.6	120.9	170.1	0.61	43	138.19	100.53	20.37	2241	10788
English Electric 1	28 2	60.96	89.2	0.4	43	47 96	34.89	26.07	2868	10866
New SFD	21.51	34.47	46.4	0.6	13	36.00	26.18	8.29	911	3137
Coimbatore MCC	19.8	39.33	52.2	0.46	23	33.00	24.01	15 32	1686	5842
Rotary furnace CC	24.06	38.1	50.4	0.64	11	40.10	29.17	8.93	982	2710
MCC 601	18	135.75	182.5	0.39	77	80.00	58.20	77.55	8531	19333
MCC 607	38 16	62 1	90	0.67	14	63 60	46.27	15 83	1742	3415
ETP MCC	113.64	175.2	240	0.62	59	206.59	150.28	24.92	2741	14645
			Total					515.56	56712	166022

Total savings in demand = 515.56 kVA

Total savings in demand per year at Rs.110/kVA = Rs.6,79,800/-

Cost of implementation = Rs.1,66,022

Simple payback period = 3 months

APPENDIX - 4/6

STAR MODE OPERATION OF GROSSLY UNDERLOADED MOTORS

FAN MOTOR

	Acidic	Non-acidic
Existing Conditions		
Rated capacity	45 kW	45 kW
Active power drawn	6.09 kW	7.17
Present loading	13.5%	15.9
Operating efficiency	80%	80%
Power output of motor	4.9 kW	5.7 kW
Proposed Conditions		
Rated capacity	15 kW	15 kW
Load on motors	40%	47.8%
Operating efficiency	90%	90%
Active power drawn	5.45 kW	6.34 kW
Energy Savings & Investments		
Power savings	6.09 - 5.45 = 0.64 kW	7.17 - 6.34 = 0.8 kW
Total energy savings @ 6000 hrs/year	3.840 kWh	4800 kWh
Cost savings	Rs.9750/-	Rs.12192/-
Cost of implementation	Nil	Nil

APPENDIX - 5/1

A. SPECIFICATION OF WASTE HEAT BOILER

Make	=	SA Babcock Belgium, N A
Type	=	La Mont Forced Circulation type
Heat source	=	SO_2 Gas from Roaster
Gas In/out	=	960 °C /350 °C
Flow of gases	=	Horizontal
Heat duty capacity	=	5.2×10^6 kcal/hr
Feed water temperature	=	105 °C
Set Steam temperature	=	254 °C
Design steam production	=	10.5 t/hr
Design steam pressure	=	42 kg/cm^2
Total heat transfer area	=	520 m^2
Total evaporation bundles	=	4 nos.

Bundles Area & Water Volume Details

Bundle No.	Surface area (m^2)	Water volume m^3
1	98	0.7
2	116	0.82
3	153	1.04
4	153	1.04

Roaster cooling coil heating surface	=	9 m^2
Drum material	=	B Quality 17 mm 4 STEE
Bundle material	=	ST 35.8/II

Appendix - 5/1 contd..

B. SPECIFICATION OF AUXILIARY BOILER

Make	= Western Engineering Ltd.
Type	= Package Boiler
Capacity	= 10 T/hr
Generation Pressure	= 10 kg/cm^2
Burner	= Jet type
Air Blower HP	= 30 kW
Air Blower RPM	= 2950
Rated Capacity of blower	= 11800 Nm^3/hr
Design oil pressure	= 20 kg/cm^2g

APPENDIX - 5/2

AUXILIARY BOILER RUNNING HOURS AND LDO CONSUMPTION

Month & Year	Running hours	LDO consumption (kL)	Hourly Consumption (kL/Hr)
April 94	103.30	109.50	1.060
May 94	308.30	194.0	0.629
June 94	187.30	99.00	0.528
July 94	362.00	199.00	0.549
Aug 94	59.30	45.00	0.759
Sept 94	76.15	61.00	0.801
Oct 94	190.00	119.00	0.626
Nov 94	91.30	71.00	0.778
Dec 94	132.15	111.00	0.840
Jan 95	38.30	31.00	0.809
Feb 95	93.00	62.00	0.667
Mar 95	255.00	189.00	0.741
Total	1896.1	1290.5	-
Average	158.00	107.54	0.680

APPENDIX - 5/3

CALCULATION OF THERMAL EFFICIENCY OF BOILER

I. BASIC DATA

Fuel in use		=	LDO
a.	Steam pressure	=	4.5 kg/cm^2g
b.	Exit flue gas temp. (T_F)	=	172 °C
c.	Diameter of combustion air blower	=	0.41 m
d.	Average air velocity	=	23.85 m/sec
e.	Percentage opening of blower	=	75 %
f.	Average LDO flow rate	=	432 kg/hr
g.	Ambient air temperature	=	32 °C
h.	Density of air	=	1.2 kg/m^3
i.	Moisture content in air	=	0.02 water kg/kg dry air

j. Composition of LDO

Basis : 100 kg LDO

C : % Carbon in LDO = 85.90
S : % Sulphur in LDO = 0.50
H : % Hydrogen in LDO = 13.60
T_F : Flue gas exit temp (°C)
T_A : Ambient air temp (°C)

II. DERIVED DATA

a. Volume of air through blower

$$= 23.85 \times \frac{\pi (0.41)^2}{4} \times 0.75 \times 3600$$

$= 8497.46\ Nm^3/hr$

b. Wt. of air through blower

= 8497.46 x 1.2
= 10196.96 kg/hr

c. Theoretical air requirement

$$T_a = 11.5\,C + 34.5\left(H - \frac{O}{8}\right) + 4.32\,S$$

$$= 11.5 \times 85.9 + 34.5\,(13.6) + 4.32 \times 0.50$$

$$= 14.59 \text{ kg/kg LDO}$$

d. Wt. of air required = 432 x 14.59

= 6302.88 kg/hr

e. % excess air

$$= \frac{\text{Actual air supplied} - \text{Theoretical air required}}{\text{Theoretical air}} \times 100$$

$$= \frac{10196.96 - 6302.88}{6302.88} \times 100$$

= 61.78 %

f. Estimation of actual CO_2 in flue gas

$$\%\text{ Excess air} = \left(\frac{\text{Max. } CO_2}{\text{Act. } CO_2} - 1\right) 100$$

Taking, Max. CO_2 for LDO = 15.30 % V/V

$$61.78 = \left(\frac{15.30}{\text{Act. } CO_2} - 1\right) \times 100$$

Actual CO_2 = 9.45 % ≃ 9.5 %

g. Total air supplied

$$= 14.59\left(\frac{61.78}{100} + 1\right)$$

= 23.603 kg air/kg LDO

h. Total flue gas quantity = (23.603 + 1)
= 24.603 kg/kg LDO

Appendix - 5/3 contd..

C. ESTIMATION OF HEAT LOSSES

a. Heat lost due to sensible heat

$$= \frac{100}{12(\%\ CO_2)} \times \left[\frac{C}{100} + \frac{S}{267} \right] \times 30.6 \times (T_F - T_A) \times \frac{1}{4.18}$$

$$= \frac{100}{12 \times 9.5} \times \left[\frac{85.9}{100} + \frac{0.5}{267} \right] \times 30.6 \times (172 - 32) \times \frac{1}{4.18}$$

= 773.70 kcal/kg

Percentage loss $= \frac{773.70}{10800} \times 100$

= 7.16 %

b. Heat loss due to Hydrogen in fuel

$$= \frac{9 \times (\%\ H_2)}{100} \times [\{1.88\ (T_F - T_A) + 2442\}] \times \frac{1}{4.18}$$

$$= \frac{9 \times 13.6}{100} \times \{1.88\ (172 - 32) + 2442\} \times \frac{1}{4.18}$$

= 792.14 kcal/kg

Percentage loss $= \frac{792.14}{10800} \times 100$

= 7.33 %

c. Heat loss due to Moisture in air

$$= \text{Wt. of air supplied} \times \text{Moisture content of air} \times (T_F - T_A) \times 1.88 \times \frac{1}{4.18}$$

$$= 23.603 \times 0.02 \times (172-32) \times 1.88 \times \frac{1}{4.18}$$

= 29.72 kcal/kg

Percentage loss $= \frac{29.72}{10800} \times 100 = 0.275$

Appendix - 5/3 contd..

d. Radiation and Convection Loss from boiler surface

Sl No	Section/Area	Average heat loss per unit area kcal/hr/m^2	Area m^2	Surface heat loss kcal/hr
1	Side surface	184 45	58 99	10881 50
2	Back surface (1)	565 80	2 00	1131 60
3	Back surface (2)	172 50	4 15	715 87
4	Firing surface	565 80	6 15	3479.67
	Total			16208 64

$$\text{Percentage Loss} = \frac{16208.64}{10800 \times 432} \times 100$$

$$= 0.347\ \%$$

Thermal efficiency of boiler = [100 - [(ii) +(iii) +(iv) +(v)]

= 100 - (7.16 +7.33 +0.275 + 0.347)

= 100 - 15.11

= 84.88 %

SUMMARY OF HEAT LOSSES

Sl. No.	Particulars	
1.	Heat input	100.00
2.	Heat loss as sensible heat in flue gas	7.16
3.	Heat loss due to hydrogen in fuel	7.33
4.	Heat loss due to moisture in air	0.275
5.	Radiation & Convection losses	0.347
	TOTAL LOSSES	15.11
	THERMAL EFFICIENCY	84.88

APPENDIX - 5/4

SUBSTITUTION OF L D O BY FURNACE OIL IN AUXILIARY BOILER

I. BASIC DATA

a. Present System

i.	Fuel in use	=	LDO
ii.	Average fuel consumption	=	680 l/hr
iii.	Monthly avg. operating hours	=	158.00
iv.	Efficiency of boiler	=	84.88 %
v.	Specific gravity of fuel	=	0.85
vi.	Calorific value of LDO	=	10800 kcal/kg
vii.	* Cost of LDO/kL	=	Rs.7310/-

b. Proposed System

i.	Fuel in use	=	Furnace Oil
ii.	Efficiency of boiler	=	84.88 %
iii.	Specific gravity of fuel	=	0.95
iv.	Calorific value of furnace oil	=	10200 kcal/kg
v.	* Cost of F.O./kL	=	Rs.5344/=

II. DERIVED DATA

i. Furnace oil requirement per hour

$$= \frac{680 \times 0.85 \times 10800}{10200 \times 0.95}$$

= 644.2 l/hr

ii.	Cost with LDO/hr	=	Rs.4970.8
iii.	Cost with FO/hr	=	Rs.3442.6
iv.	Differential fuel cost/hr	=	4970.8 - 3442.6
		=	Rs.1528.20

* The cost of fuels have been retained same as the differential cost comes to Rs.1966/kL as against present diff.cost of around Rs.2222/-

teri

Appendix 5/4 contd..

v.	Operating cost using Furnace oil		
	Power requirement for 50 l/h for pre-heating	=	3.5 kW
	Fuel to be pre-heated	=	644 l/h
	Power consumption	=	45 kWh
	Cost of power @ Rs.3.80/unit	=	Rs.171
vi.	Net cost savings/hr	=	1528.20 - 171
		=	Rs.1357
vii.	Annual operating hours	=	700 hrs
viii.	Annual savings	=	700 x 1357
		=	Rs.9.50 lakhs
ix.	Estimated investment	=	Rs.25.00 lakhs
x.	Simple payback period	=	$\frac{25.00}{9.50}$
		=	2.6 years

APPENDIX - 5/5

ESTIMATION OF SURPLUS STEAM GENERATION

BASIC DATA

i.	Steam pressure before PRV	=	34.0 kg/cm^2g
ii.	Steam pressure after PRV	=	10.0 kg/cm^2g
iii.	Steam generated	=	13.5 T/hr
iv.	Enthalpy of steam @ 34.0 kg/cm^2g	=	669.5 kcal/kg
v.	Enthalpy of steam @ 10.00 kg/cm^2g	=	663.7 kcal/kg
vi.	Efficiency of auxiliary boiler	=	84.88 %

DERIVED DATA

i. Quantity of heat @ 34.0 kg/cm^2g = 13500 x 669.5

= 9038250 kcal/hr

ii. Quantity of heat @ 10.0 kg/cm^2 = 8959950 kcal/hr

iii. Heat loss through PRV = 78,300 kcal/hr

iv. Heat loss in equivalent steam at 10.0 kg/cm^2g = $\frac{78,300}{663.7}$

= 117.97 kg/hr

v. Extra steam generation = 117.97 kg/hr

vi. Annual operating hours = 7200 hrs

vii. Estimated annual steam generation = 117.97 x 7200

= 849.384 T/annum

viii. Annual running hours of Auxiliary boiler = 1896.1 hrs

ix. Equivalent steam generation in Auxiliary boiler per annum = $\frac{849.384 \times 1896.1}{7200}$

= 223.683

Appendix - 5/5 contd..

x. Equivalent LDO consumption in Auxiliary boiler

$$= \frac{223.683 \times 663.7}{10800 \times 0.8488}$$

= 16.19 MT LDO

xi Equivalent LDO (kL) = 16.19/0.85

= 19.05

xii. Cost of LDO @ Rs.7310/ kL = 1.392 lakhs

xiii. Cost of investment = Rs.4.00 lakhs

xiv. Simple payback period = 2.9 years

APPENDIX - 5/6

ESTIMATION OF DEGREE OF SUPERHEAT

BASIC DATA

i.	Steam generation in Waste Heat Boiler	= 13.5 T/hr
ii.	Steam generation pressure	= 34.0 kg/cm^2g
iii.	Saturation temperature at 34.0 kg/cm^2g	= 241.2 °C
vi.	Steam pressure after PRV	= 10.0 kg/cm^2g
v.	Saturation temperature at 10.0 kg/cm^2g	= 183.20 °C
vi.	Enthalpy of steam at 34.0 kg/cm^2g	= 669.5 kcal/kg
vii.	Enthalpy of steam at 10.0 kg/cm^2g	= 663.7 kcal/kg
viii.	Specfic heat of steam at 10.0 kg/cm2g	= 0.47 kcal/kg°C

DERIVED DATA

i. Heat loss across PRV = 13.5 x 1000 x(669.5-663.7)

= 78,300 kcal/kg

ii. Temp. of superheated steam = Q = in m C_p (T_2-T_1)

78,300 = 13.5 x1000 x 0.47x$(T_s-183.2)$

T_2 = 183.2 + 12.34

= 195.54 °C

iii. Degree of superheat = 12.34 °C

APPENDIX - 6/1

SURVEY OF UNINSULATED PIPES, FLANGES & VALVES

Surface heat loss estimation is given by the expression

$$= \left[10 + \frac{(T_s - T_a)}{20} \right] (T_s - T_a) \ldots\ldots kcal/hr/m^2$$

Where T_s = Surface temperature (°C)

T^a = Ambient air temperature (°C)

Sl. No.	Area/Section	P/F/V	Nos.	Size	Eq.Length m	Surface area (m^2)	Surface temp. (°C)	Heat loss kcal/hr/m^2	Heat loss kcal/hr	Heat loss after insulation
	ROASTER PLANT									
1.	Ist Floor, near blower	F	1	6"	0.3	0.1436	160	2031.2	291.6	28 17
2.	Waste heat boiler, III Floor	V F	4 1	1 ½" 3.0"	4.0 0.3	0.478 0.072	120 190	1211.25 2751.20	578.9 198.08	93.78 14.12
3.	Near boiler drum control valve	V	3	1 ½"	3.0	0.359	200	3011.25	1084.4	70.43
4.	Boiler feed water pump	F	1	4"	0.3	0.096	192	2802.40	269 0	18.83
5.	Circulation water pump	F	1	6"	0 3	0.0457	200	3011.25	137.6	8.96
6.	Top floor of Roaster plant	F V V	1 1 1	3" 2" 3"	0.3 1 0 1.0	0.072 0.1595 0.239	204 214 190	3118.0 2118.0 2751 2	224.49 497.32 657.36	14.12 31.29 46.89
	LEACHING PLANT									
7.	Steam main before PRV	V	2	4"	2.0	0.64	110	1031.2	659.96	125.56
8.4	Steam main	F	1	4"	0.3	0.095	110	1031.2	97.964	18.64
9.	Steam line to Neutral pachuca's	V	6	2 ½"	6.0	1.194	140	1601.25	1911.9	234.26
	TAIL GAS TREATMENT PLANT									
10.	Main steam line to decomposition tanks	V F V	3 1 1	1 ½" ¾ 1"	3.0 0.3 1 0	0.3590 0.0179 0.0797	140 140 105	1601.25 1601.25 945.0	574.50 28 66 75.31	70 43 3.51 15.64
	Total								7287.04	794.63

Appendix - 6/1 contd..

i.	Average surface temp. after insulation (assumed)	= 50 °C
ii.	Estimated heat loss after insulation	$= \left(10 + \frac{\{50-32\}}{20}\right) \times \{50 - 32\}$
		= 196.2 kcal/hr/m^2
iii.	Total reduction in heat loss	= 7287.04 - 794.65
		= 6492.39 kcal/hr
iv.	Efficiency of Auxiliary boiler	= 84.88%
v.	Equivalent LDO in Auxiliary boiler	$= \frac{6492.39}{10800 \times 0.8488}$
		= 0.708 kg/hr
vi.	Annual auxiliary boiler running hours	= 1896.1
vii.	Annual LDO savings	= (0.708 x 1896.1)/1000
		= 1.33 MT
		= 1.33/0.85
		= 1.565 kL
viii.	Annual cost savings @ Rs.7310/- per kL	= Rs.11,440
ix.	Cost of implementation	= Rs.18,000
x.	Simple payback period	= 1.6 years

APPENDIX - 6/2

SOURCES OF STEAM LEAKAGES

Sl. No.	Source of leakage	Nos.	Plume length (m)	Estimated quantity (kg/hr)
1.	**ROASTER PLANT**			
	Top floor valve gland	1	0.6	8.8
2.	**LEACHING & PURIFICATION**			
	Main line to zinc pilot plant	1	0.6	8.8
3.	**TAIL GAS TREATMENT PLANT**			
	Control valve of Decomposition tank - 3	1	0.5	7.0
4.	Gate valve to decomposition tank 3	1	0.3	5.0
5.	Outlet of drain valve	1	1.0	16.0
	Total			45.6

i. Quantity of steam leakage = 45.6 kg/hr

ii. Annual running hours of Auxiliary boiler = 1896.1

iii. Effciency of boiler = 84.88 %

iv. Potential energy savings by periodic plugging of leakages

$$= \frac{45.6 \times 663.7 \times 1896.1}{0.8488 \times 10800 \times 1000}$$

= 6.26 MT LDO

Appendix - 6/2 contd..

v.	Equivalent LDO consumption in Auxiliary boiler	= 7.36 kL
vi.	Annual cost savings at Rs.7310/kL	= 7.36 x 7310
		= Rs.63290
vii.	Cost of implementation	= Rs.25,000
viii.	Simple payback period	= 0.40 years

APPENDIX - 7/1

ESTIMATION OF POWER RECOVERY POTENTIAL

A. BASIS

i.	Steam consumption in Neutral Pachuka's (4 nos.in operation)	= 7760 kg/hr
ii.	Steam consumption in Leaching & Purification	= 6540 kg/hr
iii.	Steam consumption in Tail Gas Treatment plant	= 200 kg/hr

Note : Depending on the steam injection in no. of Pachuka's, venting of steam takes place

iv.	Steam generated in the boiler	= 13500 kg/hr
v.	Steam pressure	= 37 kg/cm^2a
vi.	Saturated steam temperature	= 246 °C
vii.	Degree of super-heat envisaged	= 60 °C (108 °F)
viii.	Steam pressure	= 4.0 kg/cm^2
ix.	Saturated steam temperature at 4.0 kg/cm^2	= 143.63 °C
x.	Degree of super-heat envisaged	= 15 °C

Appendix - 7/1 contd..

B. DERIVED DATA

i. Saturated steam enthalpy = 2801 kJ/kg

ii. Super-heated steam enthalpy at 36 ata = 2987.8 kJ/kg

iii. Super-heated steam enthalpy at 4 ata = 2770.6 kJ/kg

Thermal Energy in kcal/kWh = $\frac{W (h_1 - h_2)}{E}$

W = kg of steam/hr

h_1 = Total heat/kg of steam at throttle kcal/kg

h_2 = Total heat/kg of steam at outlet kcal/kg

E = Alternator output kWh

$= 13500 \times \frac{\{2787.8 - 2770.6\}}{4.18}$

= 701483 kcal/h

Efficiency of Turbo alternator = 36 %

Useful output of turbine = 701483 x 0.36

= 252534 kcal/hr

Estimated power output = $\frac{252534}{860}$

= 293 kW

APPENDIX - 8/1

SPECIFICATIONS OF COMPRESSORS

A. CENTRIFUGAL COMPRESSOR - (CENTAC)

Make	=	INGERSOLL-RAND
Capacity	=	4549.65 NM^3/hr
Discharge pressure	=	7.96 kg/cm^2g
Type	=	CENTRIFUGAL
Control	=	Modulate Control

MOTOR

M/c. No.	=	185002/1
Volts, Amps	=	55.7 Amps, 6600 Volts
Rated kW	=	522
Rated RPM	=	2970

AIR DRYER

Air dryer capacity	=	50 m^3/min
Type	=	Heaterless

Appendix -8/1 contd..

B. PROCESS AIR COMPRESSORS

Make	=	KHOSLA-CREPELLE COMPRESSORS
Type	=	2HA4TER (Lubricated)
No. of Equipments	=	4
No. of Cylinders	=	4
RPM	=	750
Free Air Delivery Capacity (FAD)	=	28.3 m^3/min
Motor HP	=	248 HP
Working Pressure	=	9.27 kg/cm^2g
Oil Pressure	=	1.5 - 2.5 kg/cm^2g
Unloading Setting	=	7.5 kg/cm^2
Coupling	=	Tyre

C. INSTRUMENTATION AIR COMPRESSOR

Data	Compressor 1	Compressor 2	Compressor 3
Make	Khosla-Crepelle	Kirloskar	Kirloskar
No. of Equipments	2	1	1
Type	2HA-2-SLT	T-BTD-LM	T-BTD-LM
No. of Cylinders	2	2	2
RPM	600	750	750
Free Air Delivery (m^3/min)	7.0	8.70	9.0
Motor HP & RPM	75, 1480	100, 1500	100, 1500
Working Pressure kg/cm^2a	7.20	8.18	8.18
Unload Setting (kg/cm^2g)	7.0	7.5	7.5
Coupling	Belt	Belt	Belt

APPENDIX - 8/2

COMPRESSORS ENERGY CONSUMPTION & RUNNING HOURS

A. CENTAC COMPRESSOR POWER CONSUMPTION AND RUNNING HOUR DETAILS

Month & Year	Power consumption (kWh)	Running hour	Average power kW
April 1994	237420	533.00	445.44
May	214290	509.00	422.23
June	14310	38.00	376.58
July	181530	390.00	465.46
August	277200	780.00	355.38
September	316260	696.00	454.39
October	359010	720.00	498.62
November	305370	637.00	479.39
December	297090	658.00	451.50
January 1995	324720	694.00	467.89
February	285030	668.00	426.69
March	161100	489.00	329.45
Total	2973330	6812.0	436.48

B. RUNNING HOURS OF PROCESS AIR COMPRESSORS

Year	Running Hours			
	PAC - I	PAC - II	PAC - III	PAC - IV
1992 - 93	-	5772	5143	5234
1993 - 94	1072	4153	5052	6513
1994 - 95	1409	849	1717	-

C. RUNNING HOURS OF INSTRUMENT AIR COMPRESSORS

Year	Running Hours			
	IAC - I	IAC - II	IAC - III	IAC - IV
1992 - 93	1478	3732	4580	5783
1993 - 94	1591	3921	3633	5039
1994 - 95	118	51	7306	1156

APPENDIX - 8/3

OBSERVATIONS ON CENTAC COMPRESSOR

A. COMPRESSOR ENERGY CONSUMPTION

Date : 10.8.95

Sl. No.	Time	Energy Meter Reading	Current (Amps)
1	11 00	405.0	39.5
2	12.00	360.0	39.5
3	13.00	405.00	39.5
4	14.00	495.00	39.5
5	15.00	315.00	39.5
6	16.00	315.00	39.5

B. OBSERVATIONS ON MEASUREMENT OF AIR & WATER TEMPERATURE MEASUREMENTS

Sl No	Time	Air Temperature °C				Water Temperature °C			
		Ist Stage Intercooler		IInd Stage Intercooler		Ist stage Intercooler		IInd Stage Intercooler	
		Inlet	Outlet	Inlet	Outlet	Inlet	Outlet	Inlet	Outlet
1	10 00	42 7	-	78	-	24	34	24	34
2	10 30	43 3	-	78	-	22	34	22	34
3	11 00	43 8	-	78	-	24	34	22	34
4	11 30	43 3	-	78	-	24	34	24	35
5	12 00	43 8	-	78	-	24	34	24	36
6	13 45	43 3	-	80	-	24	34	24	36
7	14 30	43 8	-	78	-	24	36	24	37
8	Avg temp	43 4	-	78 28	-	24	34 2	23 4	35.1

Appendix - 8/3 Contd..

C. OBSERVATIONS ON MEASUREMENT OF AIR AND WATER PRESSURE

Sl No.	Time	Air Pressure kg/cm^{2}g				Water Pressure kg/cm^{2}g			
		Ist Stage Intercooler		IInd Stage Intercooler		Ist stage Intercooler		IInd Stage Intercooler	
		Inlet	Outlet	Inlet	Outlet	Inlet	Outlet	Inlet	Outlet
1	10 00	1 0	2 25	1 75	5 41	1 00	-	1 0	-
2.	10 30	1 0	2 25	1 75	5 41	1.05	-	1 0	-
3.	11 00	1 0	2 25	1.75	5 48	0 75	-	-	-
4	11.30	1 0	2 25	1.70	5 20	0 75	-	-	-
5	12 00	1 0	2 25	1 75	5.41	0 65	-	-	-
6.	13 45	1 0	2.30	1 75	5 98	0 75	-	-	-
7	14 30	1 0	2 25	1 75	5 69	0 95	-	-	-

APPENDIX - 8/4

REDUCTION OF COMPRESSED AIR CONSUMPTION IN ROASTER PLANT

BASIC DATA

i.	Average air pressure	=	6.5 bar
ii.	Present air flow inner pipe dia	=	¾" = 20 mm
iii.	Proposed air flow pipe inner dia	=	½ " = 9.30 mm
iv.	No. of hours operation/day	=	3 hrs
v.	Running hours of compressor	=	280 days
vi.	Rated FAD of compressor	=	1264 dm^3/s

DERIVED DATA

i.	Air usage through ½ " pipe dia	=	79.74 dm^3/s
ii.	Air usage through ¾ " pipe dia	=	395.0 dm^3/s
iii.	Difference in compressed air usage	=	395.0 - 79.74
		=	315.26 dm^3/s
		=	1134.93 NM^3/h
iv.	Operating FAD of compressor	=	90 % Rated FAD
		=	1264.0 x 0.9
		=	1137.6 dm^3/s
		=	4095.36 NM^3/hr
v.	Average energy consumption	=	436.48 kWh
vi.	Energy savings/hr	=	$\frac{1134.93}{4095.36} \times 436.48$
		=	120.95 kWh
vii.	Energy savings/day	=	120 kW x 3
		=	360 kWh

Appendix - 8/4 contd..

viii.	Annual operating days of compressor	=	280 days
ix.	Annual energy savings	=	360.00 x 280
		=	1,00,800 kWh
x.	Energy cost savings	=	Rs.2,56,000
xi.	Cost of implementation	=	Rs.25,000
xii.	Simple payback period	=	0.3 years

APPENDIX - 8/5

OBSERVATIONS OF PROCESS AIR COMPRESSOR

A C - III Parameter ---		P A C - I				P A C - II			P
	13.5.95	14.5.95	15.5.95	22.5.95	23.5.95	24.5.95	22.5.95	1.6.95	2.6.95

Oil pressure (kg/cm^2g)	2.8	2.7	3.0	2.3	2.4	2.2	2.4	2.5	2.4
LP 'A' Pressure (kg/cm^2g)	1.5	1.5	1.5	1.5	1.6	1 6	1.8	1.8	1.8
LP 'B' Pressure (kg/cm^2g)	2.4	2.2	2.2	1.4	1.4	1.4	1.2	1.8	1.5
After cooler pressure (kg/cm^2g)	5.0	4.6	5.4	4.2	5.4	5.4	-	-	-
Receiver pressure (kg/cm^2g)	4.8	4.4	5.4	4.2	5.7	5.4	-	-	-
Outlet water temperatures (°C)									
Intercooler (°C)	32	30	32	-	-	-	-	-	-
Aftercooler (°C)	44	44	44	-	-	-	-	-	-
Final air temp.°C	105	115	125	-	-	-	-	-	-
Current (Amps)	240	240	245	200	210	205	245	215	210

APPENDIX - 8/6

FREE AIR DELIVERY CAPACITY TEST

PROCEDURE

All compressors are designed to deliver certain cubic feet of air per minute at a specified pressure cfm of free air is the standard unit by which compressed air flow rate is measured and related to air at atmosphere. This test is conducted to confirm whether compressor is working at its rated capacity.

It is calculated by measuring the time taken to fill air receivers upto its designed pressure. By knowing receiver volume, interconnecting pipeline volume and outlet air temperature, it is possible to estimate the present FAD capacity.

CALCULATION

Volume of air receiver + inter connecting pipe lines (ft_3)	= A
Time taken to fill receiver (Minutes)	= B
Air receiver pressure (psia)	= C
Compressed air Exit temperature (°K)	= D
Inlet air Temperature (°K)	= E
Atmospheric Air Pressure (psia)	= F
Actual F.A.D of compressor (cfm)	= A x C/F x E/D x 1/B
% Deviation	$= \dfrac{\text{Rated FAD - Actual FAD}}{\text{Rated FAD}}$

APPENDIX - 9.1/1

COOLING TOWER SPECIFICATIONS

Details	Roaster & Acid Plant Cooling tower	Calcine Water Cooling tower	50 TPD Sulphuric Acid plant
Make	Roof & Loop India Ltd.	Paharpur Cooling Tower	Paharpur Cooling Tower
Type	Induced Draft	Induced Draft	Induced Draft
Year of Manufacture	1995	-	-
Capacity (Nm^3/hr)	2000	100	155
No of cells	2	2	-
No. of fans	2	1	-
No of blades	8	6	-
Dimension (L x W x H) m	15 x 8 5 x 16	-	
Design wet bulb temperature (° C)	27 7	-	-
Type of Blades	Aluminium	Aluminium	Aluminium
Rated kW of fan	45	7 5	-
Design water inlet temp (°C)	-	-	37
Design water outlet temp (°C)	-	-	32

TATA ENERGY RESEARCH INSTITUTE
BANGALORE

APPENDIX - 9.1/2

COOLING TOWER PERFORMANCE DETAILS

A. OBSERVATION OF COOLING WATER, DRY BULB & WET BULB TEMPERATURES, RANGE & APPROACH

1. ROASTER & ACID PLANT COOLING TOWER

CELL NO.1

Sl. No	Time (Hrs)	Cooling water Inlet temp °C	Cooling water Outlet temp °C	Dry Bulb °C	Wet Bulb °C	Range °C	Approach °C	No of pumps	Fan
1	9 00	43	34 5	30.5	26 5	8 5	8 0	1	W
2	10 30	43	36 5	32 5	27 0	6 5	9 5	2	W
3	12 00	44	36 5	35 5	28 0	7 5	8 5	2	W
4	14 30	43	32 5	33 0	28 0	10 5	4 5	2	W
5	16 30	43	32 0	33.5	28 5	11 0	3 5	2	W
6	Average	43 20	34 4	33.0	27 6	8 8	6 8	-	-

CELL NO.2

Sl. No.	Time (Hrs)	Cooling water Inlet temp.°C	Cooling water Outlet temp.°C	Dry Bulb °C	Wet Bulb °C	Range °C	Approach °C	No. of pumps	Fan N/NW
1.	9.00	45	34.5	30.5	26.5	10.5	8.0	2	N
2.	10.30	44	34.0	32.5	27.0	10.0	7.0	1	W
3.	12.00	44	35.5	35.5	28.0	8.5	7.5	1	W
4.	14.30	43	33.0	33.0	28.0	10.0	5.0	1	W
5.	16.30	40	33.0	33.5	28.5	7.0	4.5	1	W
6.	Average	43.2	34.0	33.0	27.6	9.2	6.4	-	-

W - Working

2. CALCINE COOLING WATER COOLING TOWER

Date : 28.7.95

Sl No	Time (Hrs)	Cooling water Inlet temp °C	Cooling water Outlet temp °C	Dry Bulb °C	Wet Bulb °C	Range °C	Approach °C	No of pumps	Fan N/NW
1	9 00	28 5	26 0	29 0	26 0	2 5	0 0	1	W
2	11 00	29 5	27 0	29 5	26 5	2 5	0 5	1	W
3	13 30	31 0	28 0	32 0	26 0	3 0	2 0	1	W
4	15 30	31 0	29 0	33 0	28 5	2 0	0 5	1	W

Appendix - 9.1/2 contd..

B. COOLING TOWER PUMP DETAILS

Sl. No.	Pump Details	Rated kW	Design Parameters		Operating Parameters			
			Flow rate m^3/hr	Head (m)	Flow rate m^3/hr	Head m	Amps	% Loading
	ROASTER & ACID PLANT							
1.	Acidic Pump	110	550	60	-	42	150	45.8
2.	Acidic Pump	110	550	60	-	-	-	-
3.	Non Acidic Pump	110	550	60	-	38	150	77.5
4.	Non Acidic Pump	110	550	60	-	38	150	59.5
	CALCINE COOLING WATER							
1.	Hot well	18.5	-	-	-	-	-	69.7
2.	Cold well	30.0	155	26.5	158.3	25.0	40	70.5

APPENDIX - 9.1/3

ROASTER PLANT COOLING WATER ANALYSIS

Date	Parameter	Cooling water	Non-Acidic water
2.8.95	pH	8.2	8.2
	Total hardness ($CaCO_3$)	112	108
	Total soluble solids	< 10 mg	< 10 mg
	TDS	134	154
3.8.95	pH	7.6	7.8
	TDS	136	232
	TSS	< 10	< 10
6.8.95	pH	8.1	8.0
	TDS	160	175
8.8.95	pH	8.1	8.1
	TDS	170	170

APPENDIX - 9.2/1

MAJOR USERS OF COOLING WATER IN THE SINTERING SECTION

Lead Smelter Cooling Towers 1 & 2

Section	Equipment	Pr required kg/cm^2g	Quantity reqd. l/h
a Charge preparation	Drum mixer no 1	1 0	500
b. Sintering section	Drum mixer no 2	4.0	500
	Bearing cooling of RC fan	4 0	500
	Vertical jacket cooling	4 0	1000
	Horizontal jacket cooling	4 0	1000
c. Crusher section	Smooth roller crusher bearing	4.0	500
	Drum cooler	4 0	600
d. Blast furnace	Jackets	2.0 }	
	Top cooling	2 0 }	153340
	Mould cooling	2 0 }	
e. Gas cleaning	PHE slurry cooling	2 5	

APPENDIX - 9.2/2

PERFORMANCE EVALUATION OF COOLING TOWERS - LEAD PLANT

Cooling Tower No.1

Particulars	Time (Hours)			
	09.00	10.45	11.45	14.00
Dry bulb temp. °C	31.5	34.5	34.0	34.5
Wet bulb temp. °C	29.0	29.0	29.0	29.0
Cooling water inlet temp °C	38.0	42.0	42.5	44.0
Cooling water Outlet temp °C	36.0	38.0	38.5	39.0
Makeup water temp. °C	35.0	35.5	35.0	36.0
Water outlet temp. after adding makeup water °C	34.5	36.0	36.5	37.0
Range * °C	2.0	4.0	4.0	5.0
Approach ** °C	7.0	9.0	9.5	10.0
Efficiency % * **	**28.5**	**44.0**	**42.0**	**50.0**

* Range = Inlet - Outlet

** Approach = Outlet - WBT

*** Efficiency = $\frac{\text{Range}}{\text{Approach}} \times 100$

Cooling Tower No.2

Particulars	Time (Hours)			
	09.00	10.45	11.45	14.00
Dry bulb temp. °C	31.5	34.5	34.0	34.5
Wet bulb temp. °C	29.0	29.0	29.0	29.0
Cooling water inlet temp °C	40.0	44.5	43.5	45.0
Cooling water Outlet temp °C	39.0	43.0	42.5	43.5
Makeup water temp. °C	35.0	35.5	35.0	36.0
Water outlet temp. after adding makeup water °C	38.0	42.5	42.5	43.5
Range °C	1.0	1.5	1.0	1.5
Approach °C	10.0	14.0	13.5	14.5
Efficiency %	**10.0**	**10.70**	**7.40**	**10.34**

Appendix - 9.2/2 contd..

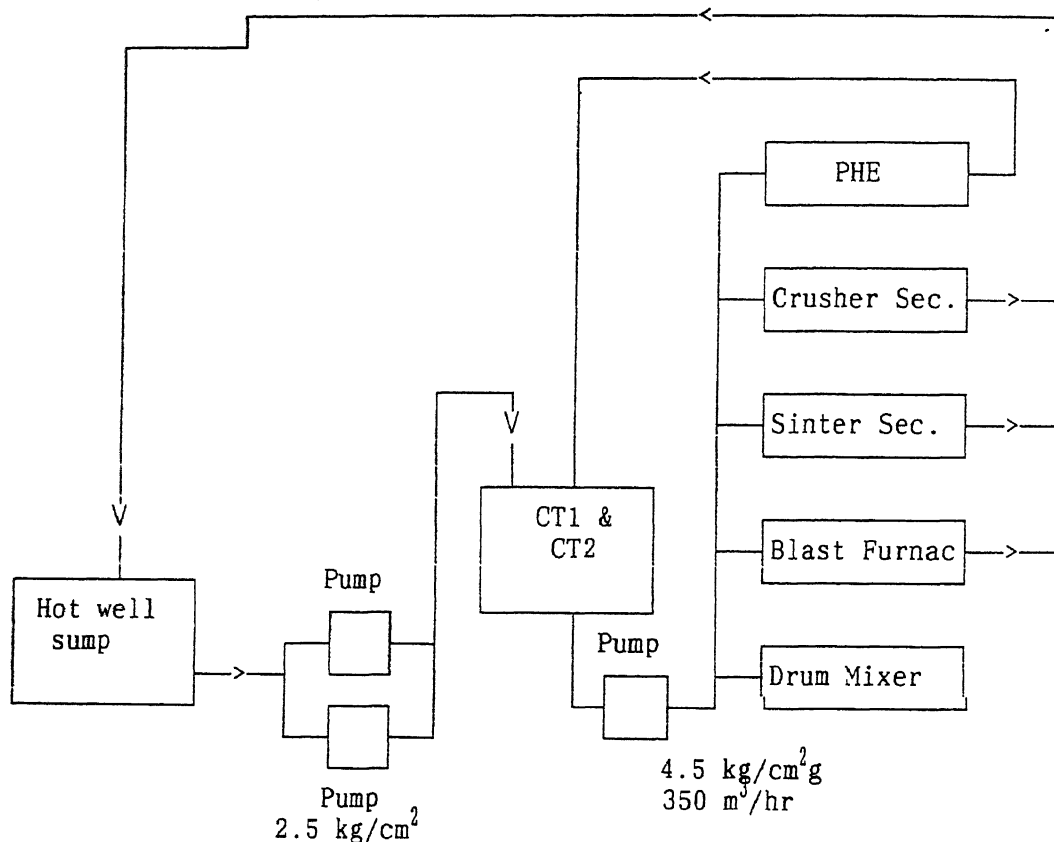
EFFICIENCY EVALUATION OF COOLING TOWER -
LEAD REFINERY PLANT

Particulars	Time (Hours)			
	10.00	11.00	12.00	14.00
Dry bulb temp. °C	33.0	33.5	34.0	35.5
Wet bulb temp. °C	27.5	29.0	29.5	29.0
Cooling water inlet temp °C	33.0	33.5	35.5	38.0
Cooling water Outlet temp °C	31.0	32.0	33.0	34.5
Range °C	2.0	2.0	2.5	3.5
Approach °C	3.5	3.0	3.5	5.5
Efficiency %	57.14	66.67	71.42	63.63

APPENDIX - 9.2/3

ONCE THROUGH SYSTEM OF COOLING WATER CIRCUIT - LEAD SMELTER PLANT

In the present system, the water is pumped at 4.5 kg/cm^2 to user ends and the return water collected in a hot well sump. Two pumps are operated to pump the hot water to the top of cooling towers.

Present System

CT1 - CT2 - Cooling Tower 1 & 2

Data

Total cooling water flow rate	=	350 m^3/hr
Cooling water pressure	=	4.5 kg/cm^{2}g
No.of pumps operated to pump cooling water to userends	=	1

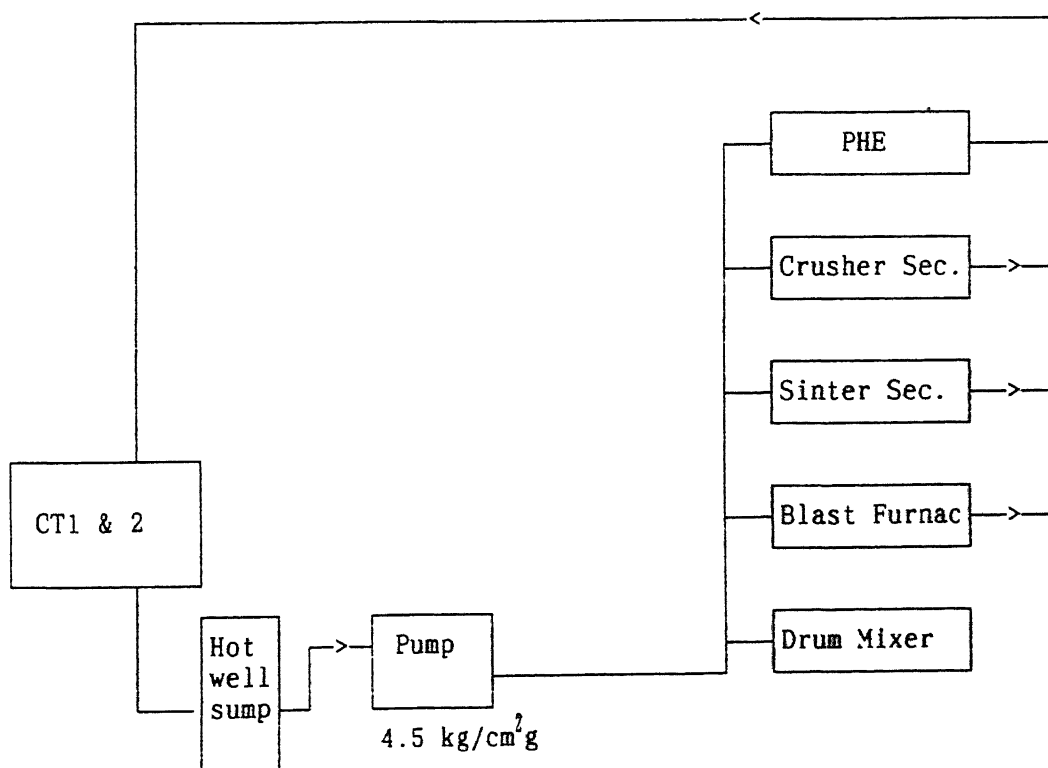
Appendix - 9.2/3 contd..

Pump power consumption	= 75.8 kW
No.of pumps operated to pump hot water from sump	= 2

Power consumption :

Pump 1	= 9.21 kW
Pump 2	= 8.58 kW
Total power consumption	= 17.79 kW

Recommendation

In all user ends, return water pressure is sufficient enough due to elevation of user equipment and high supply water pressure. This adequate head of return line enable the water to fall directly on the top of cooling water. This once through system avoids the two hot well pumps. The existing hot well can be used as cold well from which water can be pumped to utilisation areas.

Appendix - 9.2/3 contd..

To implement this measure the existing return pipe has to be replaced with 14 inch pipe, and structure has to be made for the support. The total length of pipe required is estimated at 80 mts.

The existing sump capacity is 125 m^3 (hot well and cold well).

Additional sump capacity of 140 m^3 is required in order to avoid over flow of cooling water during power failure.

Savings

Savings in power	= 17.79 kW
Savings in energy	= 17.79 x 330 x 24
	= 140896 kWh/year
Cost savings	= Rs.5.35 lakh/year
Investment	
- Piping cost	= Rs.2.00 lakh
- Structural cost	= Rs.2.00 lakh
- Sump	= Rs.4.00 lakh
Total investment	= Rs.8.00 lakh
Simple payback period	= 1.50 years

APPENDIX - 9.3/1

COOLING TOWER DETAILS - D G POWER HOUSE

A. COOLING TOWER SPECIFICATION

Details	DG Power House
Make	Paharpur Cooling Tower
Type	Induced Draft
Year of Manufacture	1989
Capacity (Nm^3/hr)	600
No.of cells	2
No. of fans	2
Design wet bulb temperature (° C)	28.8
Type of Blades	Aluminium
Rated kW of fan	22.0
Design water inlet temp. (°C)	45
Design water outlet temp. (°C)	30

B. OBSERVATIONS ON COOLING WATER INLET & OUTLET TEMPERATURES, DRY BULB, WET BULB, RANGE & APPROACH

Sl. No	Time (Hrs)	Cooling water Inlet temp °C	Cooling water Outlet temp °C	Dry Bulb °C	Wet Bulb °C	Range °C	Approach °C	No. of pumps	Fan W/NW
1	10 30	48	32 0	28 5	27 0	16.0	5 0	2	W
2.	11 15	48	32.0	28 5	27 0	16 0	5 0	2	W
3	12 00	50	36 0	29 5	27.0	14 0	9 0	2	W
4	14 00	48	31 0	28 5	26 0	17 0	5 0	2	W
5	15 00	47	31 0	30 0	27.0	16 0	4.0	2	W
6	16 00	50	32 0	28 5	27 0	18 0	5 0	2	W
7.	Average	48.5	32 3	28 9	26 8	16.1	5.5	-	-

APPENDIX - 10/1

LOADING PATTERN OF PUMPS, FANS AND BLOWERS

Sl. No.	Application	Rated kW	Actual kW	% Loading
	ROASTER PLANT			
1.	Calcine slurry pump	37	22.59	61.1
2.	Cooling tower			
	Process water pump 1	110	85.2	75.5
	Process water pump 4	110	65.4	59.5
	Process water pump 3	110	50.4	45.8
3.	Calcine cold water pump	30	21.15	70.5
4.	Calcine hot water pump	18.5	12.90	69.7
	ACID PLANT			
1.	DT Pump (200 TPD)	55	52.5	96.5
2.	AT Pump (200 TPD)	75	44.37	59.16
	PUMP HOUSE			
1.	Filter water pump	75	33.39	44.5
2.	Emergency water pump	110	52.5	47.7
3.	Filter water pump	37	36.81	98.16
4.	Rectifier water pump	37	19.8	52.8
5.	Clarified water pump	75	51.03	68.04
6.	Clarified water pump	75	48.0	64.0
	CELL HOUSE			
1.	Electrolyte pump 82	37	27.96	74.56
2.	Electrolyte pump 83	37	28.56	76.16
3.	Electrolyte pump 85	37	29.1	77.60
4.	Electrolyte pump 71	37	26.37	70.32
5.	Electrolyte pump 72	37	24.78	66.08
6.	Electrolyte pump 73	37	33.9	90.40
7.	Cold solution pump 66 b	15	8.88	59.20
8.	Electrolyte pump 48	15	8.01	53.40

Appendix - 10/1 contd..

Sl. No.	Application	Rated kW	Actual kW	% Loading
9.	Electrolyte pump 78	37	35.94	95.84
10.	Electrolyte pump 76	37	32.79	87.44
11.	Electrolyte pump 75	37	30	80.00
12.	Electrolyte pump 47	37	3.63	9.68
	LEACHING PLANT			
1.	Pachuka discharge pump 08	18.5	10.5	56.76
2.	Pachuka discharge pump 09	15	9.9	66.00
3.	Neutral Dorr pump 14 A	15	10.71	71.40
4.	Purification pump I stage 16 A	18.5	9.6	51.89
5.	Purification pump III stage 19 A	18.5	10.02	54.16
6.	Purification pump II stage 13	18.5	9.6	51.89
7.	Heat exchanger inlet	18.5	9.6	51.89
8.	Acid Dorr overflow	18.5	9.24	49.95
9.	ZnO_2 Ball mill 31 Pump	18.5	9.18	49.62
10.	ZnO_2 Ball mill 32 Pump	18.5	6.93	37.46
11.	ZnO_2 Ball mill 23 Pump	18.5	5.82	31.46
12.	Pachuka discharge pump 07	15	5.85	39.00
13.	Pachuka discharge pump 07A	15	9.36	62.40
14.	Slime leaching inlet pump	15	7.77	51.80
15.	New pachuka discharge pump	11	3.3	30.10
16.	Dorr pit pump	18.5	10.5	56.76
	SILVER FLOTATION DEPT.			
1.	Neutralisation tailing pump	15	4.74	31.60
2.	D5 over flow pit pump	11	7.05	64.09
3.	Agitator lime slurry pump 14	11	3.51	31.91
4.	Agitator lime slurry pump 15	11	5.1	46.36
5.	Dorr 4 overflow pump	11	8.94	81.27
6.	Rectifier return pump	11	3.75	34.09
7.	Filtrate pump	7.5	4.83	64.40

Appendix - 10/1 contd..

Sl. No.	Application	Rated kW	Actual kW	% Loading
	DL PLANT			
1.	Fresh air fan	37	29.01	78.41
2.	Ignition air fan	22	16.74	76.09
3.	Combustion air fan	9.3	3.06	32.90
	GAS CLEANING PLANT			
1.	Hot water sump pump 2	22	8.58	39.00
2.	Hot water sump pump 1	22	9.21	41.86
3.	Hot water dewatering pump	18.5	9.33	50.43
4.	RC pump 1B	18.5	6.06	32.76
5.	Stripper feed pump 22 A	18.5	5.7	30.81
	NEW BLAST FURNACE			
1.	Granulation pit pump 1	37	17.61	47.59
2.	Granulation pit pump 3	37	18.69	50.51
3.	Scrubber pump 3	18.5	3.75	20.27
4.	Fumes exhaust blower	22	11.7	53.18
5.	Roots blower	110	52.62	47.84
6.	Fumes exhaust blower	110	63.6	57.82
	COOLING TOWER			
1.	Cooling tower pump 1	75	75.6	100.80
2.	Return water pump 2	5.5	3	54.55
3.	Cooling tower pump 2	75	66.9	89.20
4.	Cooling tower pump 3	37	18.45	49.86
	LEAD REFINERY			
1.	Kettle Blower 2	18.5	10.56	57.08
2.	Kettle Blower 5	18.5	8.76	47.35
3.	Kettle Blower 6	18.5	7.53	40.70
4.	Kettle Blower 7	18.5	12	64.86
5.	Kettle Blower 8	18.5	8.7	47.03

Appendix - 10/1 contd..

Sl. No.	Application	Rated kW	Actual kW	% Loading
	EFFLUENT TREATMENT PLANT			
1.	Horizontal pump 3	11	5.94	54.00
2.	Horizontal pump 1	11	3.9	35.45
3.	Lime Agitator pump 2	5.5	2.07	37.64
4.	Lime Agitator pump 1	5.5	1.95	35.45
5.	F D Pump 3	30	8.73	29.10
6.	Air blower 1	15	5.82	38.8
	HT MOTORS			
1.	RC Gas fan	250 250	172.5 147.0	69 59
2.	SO2 Blower (200 TPD)	500	279	56
3.	SO2 Blower (50 TPD)	380	190.5	50
4.	Bag House Blower	262	88	34

A. SPECIFICATIONS OF ROASTER

Make	=	LURGI CHEMEC UND HUTTENTECHNICK, GERMANY
Type	=	Fluidised bed
Roaster volume	=	720 m^3
Hearth area	=	35 m^2
Design furnace bed temperature	=	900 - 950 °C
Material feed rate (Dry basis)	=	10 - 11 T/hr
Maximum air flow rate	=	29,520 m^3/hr
Operating air flow rate	=	18,000 - 19,000 Nm^3/hr
No. of nozzles	=	3550

TYPE OF REFRACTORIES

Refractory	Thickness (mm)
Fire clay brick	230
Insulation brick	100
Hysil brick	50

Appendix - 11/1 contd..

B. FEATURES OF WASTE HEAT BOILER

i.	Type	=	La Mont Forced circulation type
ii.	Designers	=	Babcock, Belgium
iii.	Flow of gases	=	Horizontal
iv.	Heat duty capacity	=	5.2×10^6 kcal/hr
v.	Steam production	=	About 10.5 MT saturated steam at 42 kg/cm^2g
vi.	Total heating surface	=	520 m^2
vii.	Exit temp.of flue gases	=	350 °C

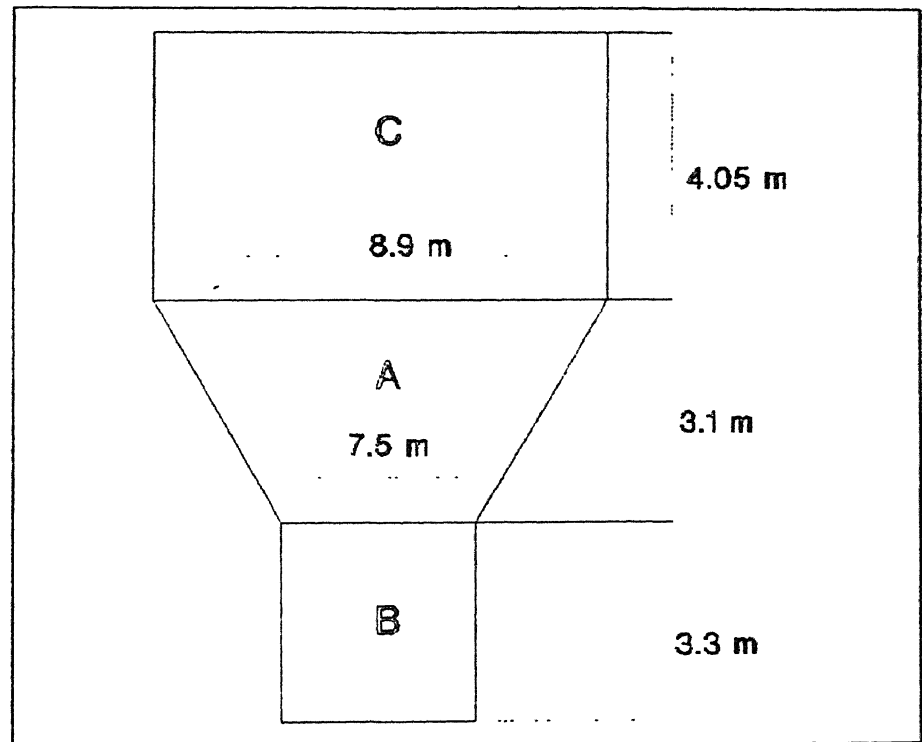
APPENDIX - 11/2

MONTHWISE DETAILS OF ZINC CONC. ROASTED, LDO CONSUMPTION, RUNNING HOURS OF ROASTER, PREHEATING BURNERS AND LANCERS

Month & Year	Zinc Conc. (MT)	LDO consumption (kL)	Running hours	Pre-heating Hours		
				Burners	Lancers	Total
April 94	3276 00	12 0	336 58	74 00	6.00	80 00
May 94	2906 00	12 0	291.66	60 00	5 30	65 30
June 94	4458 81	11 0	434 33	88 00	2 30	90 30
July 94	3282 19	9 5	335 50	92 00	5 45	97 45
Aug 95	6909 00	7 0	666 25	12 00	4 45	16 45
Sept 94	6724.00	8.5	633 08	13 00	2.30	15 30
Oct 94	5568 00	10 0	524 16	53 00	3 30	56 30
Nov 94	6157 00	8.0	594 08	30 30	6 00	36 30
Dec 94	5902 00	15 0	572 75	68 00	9 00	77 00
Jan 95	7539 00	8 0	693 33	-	-	-
Feb 95	5988 00	9 0	564 00	40 00	3 00	43 00
Mar 95	4302 00	10 0	399 66	63 00	6 30	69 30
Total	63012 00	120 00	6045 38	593 30	53 40	646 70

APPENDIX - 11/3

QUANTIFICATION OF SURFACE HEAT LOSSES OF ROASTER

I. ROASTER EQUIPMENT SECTIONS AND DIMENSIONS

NOTATIONS :

1. Surface Towards DM water tank
2. Surface towards WHB side
3. Surface opposite 1
4. Surface opposite 2

Appendix - 11/3 contd..

II. OBSERVED PARAMETERS

Surface heat loss estimation is given by the expression

$$= \left[10 + \frac{(T_s - T_a)}{20} \right] (T_s - T_a) \ldots\ldots \text{kcal/hr/m}^2$$

Where T_s = Surface temperature (°C)

T^a = Ambient air temperature (°C)

Sl No	Area/Section	Avg surface temp. (°C)	Heat loss kcal/hr/m²	Area (m²)	Heat loss kcal/hr
1.	**B Section**				
	Between 1 & 2	54 0	244.20	9 70	2379 2
		71 0	466.05	9.70	4521 6
	Between 1 & 4	76 0	536 80	9 70	5208 0
		71 0	466.05	9 70	4521 6
	Between 1 & 3	92 0	780 00	9 70	7567 56
	(Material feeding)	69 0	438 45	9 70	4253 80
	Between 3 & 2	81 0	610 05	9 70	59 18 7
		68 0	424 80	9 70	4121.40
					38551 86
2	**A Section**				
	Between 1 & 2	81 0	610 00	9.167	5591.87
		84 0	655 20	8 693	5695.78
		88 0	716.80	8.209	5884.20
	Between 1 & 4	84 0	655.20	9.167	6006.22
		77 0	551.20	8.693	4791.69
		92 0	780.00	8.209	6403 09
	Between 1 & 3 *	-	1982 00 *	26.069*	17171 85
	Between 3 & 2	-	1986 40 *	26.069*	17201 00
	Total				68745 7

* **Computed values**

Appendix - 11/3 contd..

Sl No	Area/Section	Avg. surface temp (°C)	Heat loss kcal/hr/m²	Area (m²)	Heat loss kcal/hr
3.	C Section				
	Between 1 & 2	98.00	877 80	8 90	7812.42
		80.00	595 20	8.90	5297.28
		98 00	877 80	8.90	7812.42
		80 00	595.20	8 90	5297.28
	Between 2 & 3	90.00	748.20	8 90	6658 98
		77 00	551 25	8 90	4906 12
		90 00	748 20	8 90	6658.98
		77 00	551 25	8 90	4906 12
	Between 3 & 4	98 00	877 80	8 90	7812 42
		103 00	962 05	8 90	8562 24
		98 00	877.80	8 90	7812 42
		103 00	962 05	8 90	8562 24
	Between 1 & 3				23130 2*
	Total				105229.12

Total heat loss = 38551.86 + 68745.7 + 105229.12
= 212526.68 kcal/hr

SUMMARY OF HEAT LOSSES

Roaster section	Area (m²)	Heat loss kcal/hr	% of Total heat loss	% of Total area
B Section	77.600	38551.86	18.14	26.88
A Section	104.276	68745.70	32.36	36.12
C Section	106.80	105229.12	49.52	37.00
Total		212526.68	100	100.00

APPENDIX - 11/4

DESIGN AND OBSERVED TEMPERATURES AT VARIOUS PROCESS EQUIPMENTS

Sl. No	Area/Equipment	Design Temperature (°C) conditions			Actual Temperature (°C) conditions		
		Inlet	Outlet	Δ T	Inlet	Outlet	Δ T
1	Furnace	32-35	900-950	868-915	32	965	933
2	Boiler Bundle I	900-950	650	250-300	965	610	355
	II	-	-	-	610	-	-
	III	-	-	-	-	-	-
	IV	-	350	-	-	360	-
3	Cyclone Separator	350	330	20	360	327	33
4	Hot gas precipitator	330	300-330	0-30	321	296	25
5	Scrubber	300-330	67	233-263	296	57	239
6	Stand pipe	67	53	14	-	-	-
7	WGP- I	53	53	0	-	-	-
8	Star cooling	53	38	15	-	-	-
9	WGP-II	38	38	0	-	42	-

APPENDIX - 11/5

DESIGN AND OBSERVED PRESSURES AT VARIOUS PROCESS EQUIPMENTS

Date : 31.7.95

Sl No.	Area/Equipment	Design Pressure (mmwg)			Actual Pressure (mmwg)		
		Inlet	Outlet	Drop	Inlet	Outlet	Drop
1	Roaster	1700	± 00	1700	± 1700	± 0	- 1700
2	Waste Heat Boiler	-	- 40	40	± 0	- 32	- 32
3	Cyclone Separator	- 40	- 140	100	- 32	- 80	- 48
4	Hot gas precipitator	- 140	- 170	30	- 80	- 100	- 20
5	Blower	- 170	± 0	170	- 100	+ 200	+ 300
6	Scrubber	± 0	- 30	30	+ 200-	+ 180	- 20
7	Stand Pipe	- 30	- 120	90	+ 180	+ 10	- 170
8	Wet Gas precipitator - I	- 120	- 150	30	+ 10	- 25	- 35
9	Star Cooling	- 150	- 250	100	- 25	- 105	80
10	Wet Gas Precipitator - II With Fan Without Fan	 - 250 - 250	 - 280 - 450	 30 200	- 105	- 220	115

APPENDIX - 11/6

SURFACE HEAT LOSSES FROM WASTE HEAT BOILER

Date :

OBSERVED SURFACE TEMPERATURE AND SURFACE HEAT LOSSES

Sl. No.	Area/Section	Avg Surface temp (°C)	Heat loss kcal/hr/m²	Area m²	Heat loss kcal/hr
1	3rd floor	58	256 45	10 80	2769 6
	2nd floor (DM Water side)	65	345 00	11.90	4105 5
		62	306 45	7 10	2175.7
		61	293 80	4 68	1314.9
		78	522.45	5 12	2674 9
		82	531.05	3.06	1625.0
				42.66	14665 6
2.	Coil section	60	358 05	2.40	859 32
		61	293 80	2 60	781 50
		62	306 45	1 59	487.25
				6 59	2128 07
3	Ist Floor	62	306 45	7 38	2261 60
		60	281 25	7 38	2075 62
		69	397.80	6 03	2398 73
	Doors	73	452 2	1 35	610 47
				22 14	7346 42
4	Ist Floor opposite side	62	206 45	7 38	2261 60
		60	281.25	7 38	2075 62
		65	345.00	6 03	2080 35
	Doors	84	610 05	1 35	823 56
				22 14	7241 13

Above observed losses are from one side surface

Side surface heat loss = 14665.6 x 2 + 2128.07 x 2 + 7346.42 + 7241.13

= 48174.89 kcal/hr

Taking loss through duct as around 5 %

= 48174.89 x 0.05

= 2408.74 kcal/hr

Total losses = 50583.63 kcal/hr

APPENDIX - 12/1

200 TPD SULPHURIC ACID PLANT
EQUIPMENT FEATURES

A. DESIGN CONVERTER BED TEMPERATURES

		Inlet °C	Outlet °C
I	Bed	420	580-600
II	Bed	430	490
III	Bed	430	475-480
IV	Bed	420	425

B. DESIGN SO_2 BLOWER CAPACITY

Blower capacity - 35000 NM^3/hr
Outlet Pressure - 2950 mmwg

C. ABSORPTION TOWER

Circulation pump capacity - 220 m^3/hr

a. COMBUSTION FURNACE (KREBS)

i. Type = Cyl - horizontal

ii. Lining = Cold face -Insulating brick -110mm
Hot face insulating brick -110 mm

Brick	Thickness (mm)
Insulating brick	220
Silmonite	220
Mortor	10mm

iii. Operating pressure = 250 mmwg

iv. Operating efficiency = 85 %

v. Insulation = Hot

Appendix - 12/1 contd..

b. CENTRIFUGAL COMBUSTION AIR BLOWER

Manufacturer	=	Wesman Engg. Co. Pvt. Ltd.
Type	=	Centrifugal
Capacity	=	7200 Nm^3/hr
Pressure	=	450 mmwg
Motor (HP)	=	25

c. CENTRIFUGAL DILUTION AIR BLOWER

Manufacturer	=	Wesman Engg. Co. Pvt. Ltd.
Type	=	Centrifugal
Capacity	=	2000 Nm^3/hr
Pressure	=	280 mmwg

d. PREHEATER

Make	=	KREBS
Type	=	Verical, Cylindrical
SHELL SIDE DETAILS		
Shell dia	=	2416 mm
Shell height	=	7715 mm
Shell thickness	=	8 mm
Media	=	Air/SO_2
Temp. of medium	=	45/450 °C
TUBE SIDE DETAILS		
No. of tubes	=	367
O.D. of tubes	=	57.15 mm
Height of tubes	=	6050 mm
Operating pressure	=	100 mmwg
Medium	=	Flue gas

APPENDIX - 12/2

MONTHWISE PRODUCTION DETAILS OF 200 TPD & 50 TPD SULPHURIC ACID PLANT

Month & Year	200 TPD White Acid plant (MT)	50 TPD Black Acid plant (MT)	Total Acid Production (MT)
April 94	2783.483	292.433	3075.916
May 94	2313.545	105.834	2419.379
June 94	3533.023	-	3533.023
July 94	2762.00	85.721	2847.721
Aug 94	5450.413	306.440	5756.853
Sept 94	5193.644	439.018	5632.662
Oct 94	4282.884	384.188	4667.072
Nov 94	4764.024	229.709	4993.733
Dec 94	4519.732	170.470	4690.202
Jan 95	5822.655	157.623	5980.278
Feb 95	4687.153	224.986	4912.139
Mar 95	3359.320	168.791	3528.111
Total	49471.876	2565.213	52037.089

APPENDIX - 12/3

MONTHWISE ENERGY CONSUMPTION AND RUNNING HOURS

A. 50 TPD SULPHURIC ACID PLANT RUNNING HOURS

Month & Year	Production (MT)	LDO Consumption (kL)	Plant Running Hours	Pre-heater Running hours
April 94	292 433	34.0	257.45	317 50
May 94	105 834	3 0	82 30	118.30
June 94	-	4 0	4 15	24 30
July 94	85 721	5 0	92 05	133 uu
August 94	306.440	9 0	212.00	252 45
Sept 94	439.018	8 0	393.05	456.15
Oct 94	384 188	10.0	296.00	354 15
Nov 94	229 709	5 0	213.45	281 45
Dec 94	170 470	10 0	284.30	365.45
Jan 95	157.623	10 0	194.00	235.30
Feb 95	224.986	22 0	203 00	239.45
Mar 95	168 791	5 0	141.15	156 20
Total	2565 21	125.0	2372 9	2935 70
Average /month	213 76	10 42	197.7	244 64

Appendix - 12/3 contd..

B. 200 TPD SULPHURIC ACID PLANT DETAILS

Month & Year	Production (MT)	LDO Consumption (kL)	Plant Running Hours	Pre-heater Running hours
April 94	2783.483	10.0	336 35	48 20
May 94	2313 545	11 0	291 40	48 00
June 94	3533 023	14 0	434 20	64 30
July 94	2762 000	19 0	335 30	66 15
August 94	5450 413	11 0	666.15	31 00
Sept 94	5193.644	18 0	633 05	45 15
Oct 94	4282.884	15 0	524 10	63 00
Nov 94	4764.024	16 0	594 05	53 00
Dec 94	4519.732	31 0	572 45	78 30
Jan 95	5822 655	10 0	693 20	50 15
Feb 95	4687 153	11 0	564 00	50 45
Mar 95	3359 320	15 0	399 40	71 15
Total	49471 88	181 0	6043 65	668 85
Average /month	4122 65	15 08	503 62	55 73

APPENDIX - 12/4

HEAT RECOVERY FROM 200 TPD H_2SO_4 PLANT PRE-HEATER

I. BASIC DATA

Fuel in use: LDO

Sl. No.	Data	Units	Quantity
1.	Capacity of fuel tank	m^3	3.0
2.	Fuel consumption	l/h	315
3.	Furnace gas temeprature	°C	600
4.	Stack gas temperature	°C	360
5.	Capacity of combustion air blower	Nm^3/hr	7300
6.	Diameter of combustion air blower	m	0.29
7.	Pressure of combustion air blower	mmwg	450
8.	Capacity of dilution air blower	m^3/hr	20000
9.	Diameter of dilution air blower	m	0.62
10.	Pressure of dilution air blower	mmwg	280
11.	Damper opening in dilution blower	%	50
12.	Density of atmospheric air	kg/m^3	1.2
13.	Specific heat of atmospheric air	kcal/kg°C	0.24

II. DERIVED DATA

i. Average velocity of air through combustion air blower = 15.52 m/s

ii. Average volume of combustion air

$$= 15.52 \times \pi \times \frac{(0.29)^2}{4} \times 3600$$

= 3687.5 m^3/hr

= 4425 kg/hr

iii. Average compressed air flow through burners = 10% of combustion air flow

= 4425 x 0.1
= 442.5 kg/hr

Appendix -12/4 contd..

iv. Total mass of combustion air supplied = 4425 + 442.5

= 4867.5 kg/hr

v. Average air velocity through dilution air blower = 22.56 m/sec

vi. Avg.volume of dilution air $= 22.56 \times \frac{\pi(0.62)^2}{4} \times 3600 \times 0.5$

$= 12253.6\ Nm^3/hr$

Avg. mass of dilution air = 14704.35 kg/hr

vii. Total air flow rate = (Mass of combustion air) + (Mass of dil.air)

= 4867.5 + 14704.35

= 19571.85 kg/hr

viii. Heat available in entering gas = 19571.85 x 0.24x(600-32)

= 2668034.59 kcal/hr

ix. Heat supplied to entering SO_2 gas = 19571.85 x 0.24(600-360)

= 1127338.56 kcal/hr

x. Heat available in exhaust gas (assuming stack gas temp. around 200 °C) = 19571.85 x 0.24(360-200)

= 751559.0 kcal/hr

Appendix - 12/4 contd..

xi.	Heat required to pre-heat dilution air to 150 °C	= 14704.35 x0.24x(150-32)
		= 416427.19 kcal/hr
xii.	Equivalent LDO for pre-heating dilution air (Taking % η preheater = 0.85)	= $\frac{416427.19}{10500 \times 0.85}$
		= 46.66 l/h
xiii.	Annual running hours of pre-heater	= 668.85 h
xvi.	Reduction in LDO consumption /annum	= 46.66 x 668.85
		= 31.20 kL
xv.	Cost of LDO savings/annum (@ Rs.7310 per kL)	= 2.28 lakhs
xvi.	Cost of implementation towards pre-heater, new blower	= Rs.6,00,000
xvii.	Simple payback period	= $\frac{6,00,000}{2,28,000}$
		= 2.6 years

APPENDIX - 12/5

SUBSTITUTION OF LDO BY FURNACE OIL IN PRE-HEATER OF 200 TPD H_2SO_4 PLANT

I. BASIC DATA

a. Present System

i.	Fuel in use	=	LDO
ii.	Annual fuel consumption	=	181.0 kL
iii.	Annual running hours	=	668.85
iv.	Specific gravity of fuel	=	0.85
v.	Calorific value of LDO	=	10,500 kcal/kg
vi.	Cost of LDO/kL	=	Rs.7,310/-

b. Proposed System

i.	Fuel in use	=	Furnace Oil
ii.	Specific gravity of fuel	=	0.95
iii.	Calorific value of Furnace oil	=	10,200 kcal/kg
vi.	Cost of FO/kL	=	Rs.5,344/-

II. DERIVED DATA

i. Annual furnace oil requirements

$$= \frac{181.0 \times 0.85 \times 10500}{0.95 \times 10200}$$

= 166.71 kL

ii. Annual fuel cost with LDO = 13.23 lakhs

iii. Annual fuel cost with furnace oil = 8.92 lakhs

iv. Annual difference cost = 13.23 - 8.92
= 4.31 lakhs

c. Operating Cost Using Furnace Oil

i. Power requirement for 50 l/h for furnace oil preheating = 3.5 kW

ii. Fuel to be pre-heated = 500 l/h

iii. Total power requirement

$$= \frac{500 \times 3.5}{50}$$

= 35 kW

Appendix - 12/5 contd..

iv.	Cost of pre-heating @ Rs.2.54/unit	= Rs.88.90
v.	Annual pre-heating cost	= 88.9 x 668.85
		= Rs.59,385.20

III. SAVINGS

i.	Net savings/annum	= Rs.4,31,000 - 59,385.2
		= Rs.3,71,614
		= Rs.3.71 lakhs
ii.	Cost of implementation	= Rs.10.00 lakhs
iii.	Simple payback period	= 2.69 years

Appendix - 13/1 contd..

vii.	Quantity of steam reqd/ batch	$= \dfrac{1234800}{663.18}$
		= 1862 kg/batch
viii.	Quantity of steam/day	= 1862 x 25
		= 46550 kg/day
ix.	Avg. steam consumption	$= \dfrac{46550}{24}$
		= 1940 kg/hr
x.	Maximum steam consumption (considering 4 pachuka's in heating condition at a time)	= 1940 x 4
		= 7760 kg/hr

APPENDIX - 13/2

ESTIMATION OF INDIRECT STEAM USAGE IN LEACHING AND PURIFICATION

Basic Data

Pump discharge capacity = 90 m^3/hr

Rated power input = 18.5 kW

Actual power input = 9.6 kW

Pump and motor efficiency = 0.51

Power input = 0.51 x 9.6

= 4.896 kW

= 6.56 HP

$$\frac{\text{Flow x head}}{3960} = 6.56$$

$$\text{Flow} = \frac{6.56 \times 3960}{20 \times 3.3 \times 1.4}$$

= 281 GPM

$$= \frac{281 \times 4.5 \times 60}{1000}$$

= 75.87 m^3/hr

Heat required to raise temp. = 90x1.2x 1000x0.952x (90-57)

= 3392928 kcal/hr

Appendix - 13/2 contd..

Steam pressure	= 2.0 kg/cm^{2}g
	= 3.0 kg/cm^{2}a
Steam enthalpy	= 2724.7 kJ/kg
	= 652 kcal/kg
	= 652 - 133.54
	= 518.46 kcal/kg
Quantity of steam	= $\frac{3392928}{518.46}$
	= 6544 kg/hr or
	6540 kg/hr

APPENDIX - 13/3

ESTIMATION OF SURFACE HEAT LOSSES FROM NEUTRAL PACHUKAS

i. Volume $\pi r^2 h$ = 60 m^3

where, h = 4 m

$r^2 = \frac{60\ m^3}{\pi \times 4}$

= 4.7

r = 2.2 m

ii. Dia = 2.2 x 2 = 4.4 m

iii. Ambient temp. = 32°C

Surface temp = 42°C

Surface area = π x 4.4 x 4

= 55.3 m^2

Surface heat loss = $\{10 + (\frac{42-32}{20})\} \times (42-32)$

= 105 kcal/hr.m^2

Total heat loss = 105 x 55.3

= 5806 kcal/hr

= 7000 kcal/hr (considering 20% extra)

APPENDIX - 13/4

POSSIBLE ENERGY SAVINGS BY REDUCING EVAPORATION LOSSES FROM DORR THICKNER

Basic Data

i.	Volume of DORR thickner	= 400 m^3
ii.	Estimated height	= 3.0 m
iii.	Estimated dia	= 13 m
iv.	Temp. of electrolyte	= 70°C

Derived Data

i.	Surface heat losses	= 1700 Btu/hr ft^2
		= 4600 kcal/hr.m^2
ii.	Surface area	= 133 m^2
iii.	Total heat losses	= 4600 x 133
		= 611800 kcal/hr
iv.	Possible reduction of heat losses	= 611800 x 0.4
		= 244720 kcal/hr
v.	Possible energy savings	= $\frac{244720}{518.46}$
		= 472 kg/hr (steam)

APPENDIX - 14.1/1

MONTHLY PRODUCTION VIS-A-VIS POWER CONSUMPTION

Month	Production (MT)	Rectifier Power consumption (kWh/MT)	Power consumption (L.kWh)
April 94	224.444	-	7.5099
May 94	1416.283	3469.0	49.1309
June 94	1924.760	3437.0	66.1540
July 94	2605.995	3516.5	91.6398
Aug 94	2931.471	3457.0	101.3409
Sept 94	2578.714	3468.0	89.4298
Oct 94	2885.472	3493.0	100.7895
Nov 94	2690.751	3494.0	94.0184
Dec 94	2727.113	3504.0	95.5580
Jan 95	2957.163	3461.0	102.3474
Feb 95	2621.156	3568.0	93.5228
Mar 95	2823.446	3539.0	99.9217
Total	28386.77	-	991.36

Specific power consumption per MT production $= \frac{99136000}{28386.77}$

$= 3492$ kWh

Appendix 14.1/1 contd..

PRODUCTION VIS-A-VIS POWER CONSUMPTION OF THREE SELECTED CELLS

Days	Circuit I				Circuit II	
	Cell No.5		Cell No 17		Cell No 20	
	Prodn (MT)	Circuit power consn. (kWh)	Prodn (MT)	Circuit power consn (kWh)	Prodn. (MT)	Circuit power consn (kWh)
28 7.95	3 210	10241	2 830	10241	2 720	10359
29 7 95	3 200	10341	2.965	10341	2 540	10371
30 7 95	3 100	8718	2 910	8718	2 630	9618
31 7 95	2 840	8933	2 400	9052	2.650	10425
01.8 95	2 860	10118	2.620	10118	2.730	9347
02.8.95	3.135	9921	3 000	9921	2.230	9506
03.8 95	3.060	10640	2 965	10641	2.542	9976

SPECIFIC POWER CONSUMPTION

Days	Cell No.5 kWh/MT	Cell No.17 kWh/MT	Cell No.20 kWh/MT
28.7.95	3190	3619	3808
29.7.95	3232	3488	4083
30.7.95	2812	2996	3657
31.7.95	3145	3722	3934
01.8.95	3538	3862	3527
02.8.95	3165	3307	4263
03.8.95	3477	3589	3924

APPENDIX 14.1/2

MEASUREMENT OF INDIVIDUAL CELL VOLTAGES

CIRCUIT : X22

Date : 2.8.95

Time :10.00 - 12.00 Hrs

Cascade No	A1	A2	A3	A4	A5	B5	B4	B3	B2	B1	Total	Bus to bus voltage
1	U N D E R		M A I N T E N A N C E									
2	3 43	3 19	3 14	3 10	3 12	3 19	3 15	3 11	3 14	3 16	31 73	32 43
3	3 06	3 15	3 15	3 11	3 09	3 19	3 30	3 16	3 09	3 10	31 35	31 54
4	3 10	3 17	3 18	3.13	3 11	3 17	3 14	3 21	3 16	3 18	31 55	31 97
5	3 10	3 46	3 17	3 12	3 11	3 13	3 23	3 18	3 14	3 16	31 80	32 15
6	3 14	3 15	3 20	3 21	3 18	3 10	3 04	3 51	3 12	3 29	31.94	32.54
7	3 25	3 18	3 46	3.02	3 17	3 46	3 26	3 35	3 16	3 20	32 51	32 95
8	3 32	3 34	2 99	3.14	3 08	3 29	3.34	3 20	3 12	3 20	32.02	32 51
9	3 12	3 10	3 13	3 20	3 20	3 30	3 30	3 14	3 10	3 16	31 70	32 25
10	3 15	3 10	3 11	3 19	3 11	3 08	3 14	3 11	3 16	3 15	31 30	31.76
11	3 44	3 71	3 16	3 39	3 07	3 25	3 15	3 40	3 31	3 29	33 17	35 32
12	3 05	3 38	3 18	4 56	3 37	3 18	3 22	3 16	3 12	3.16	33 38	33 92
13	3 22	3 24	3 42	3 22	3 16	3 21	3.20	3 14	3 43	3.19	32.43	32 91
14	3.02	3 19	3 46	4 48	3 17	3.36	3 27	3 51	3 53	3.53	34.52	34.92
15	3 46	3 21	3 16	3 22	3 26	3 19	3 13	3 23	3 42	3 23	32 51	32.99
16	3.18	3 29	3 20	3 46	3 13	3 17	3 27	3.22	3 20	3.08	32 20	32.77
17	3 46	3 33	3 2[illegible]	3 49	3 09	3 12	3 09	3 22	3 32	3 22	32 60	33.10
18	3.21	3 48	3 17	3.27	3 20	3.21	3 21	3.19	3 16	3.17	32.27	32.81
	Total										548.98	558.85

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Appendix 14.1/2 contd

MEASUREMENT OF INDIVIDUAL CELL VOLTAGES

CIRCUIT : X22

Date : 2.8.95

Time : 14.45 - 16.00 Hrs

Cascade No	A1	A2	A3	A4	A5	B5	B4	B3	B2	B1	Total	Bus to bus voltage
1	U N D E R	M A I N T E N A N C E										-
2	3 12	3 17	3 14	3 10	3 102	3 13	3 17	3 08	3 14	3 13	31 28	31 88
3	3 03	3 10	3 09	3 09	3 06	3 20	3 08	3 08	3 07	3 07	30 87	31 66
4	3.11	3 18	3.18	3 13	3 12	3 16	3 30	3 03	3 16	3.17	31 54	32 17
5	3 18	3 46	3 18	3 23	3.23	3 16	3 24	3 21	3 16	3 17	32 22	32 83
6	3 15	3 34	3 23	3 21	3 15	3 17	3 15	3 32	3 16	3 38	32.26	32 89
7	3 26	3 23	3 18	3.15	3 16	4 08	3.25	3 39	3 20	3 18	33 08	33 64
8	3 36	3 22	3 14	3 39	3 24	3 27	3 22	3 21	3 39	3 19	32 63	33 20
9	3 15	3 14	3 15	3 33	3 12	3 32	3 24	3 15	3 12	3 18	31 90	32 45
10	3 18	3 12	3 16	3 20	3 14	3 10	3 16	3 14	3 19	3 29	31 68	33 15
11	3 42	3 49	3 44	3 57	3 24	3 25	3 19	3 21	3 72	3 28	33 81	34 63
12	3 20	3 20	3 19	3 56	3 42	3 19	3 23	3 18	3 12	3 16	32 45	33 03
13	3 22	3 23	3.35	3 24	3 18	3 21	3.51	3 15	3 43	3 19	32 71	33 25
14	3 19	3 34	3 41	3 49	3 29	3.36	3 28	3 39	3 57	3 56	33 80	34 43
15	3.21	3 2	3.17	3 23	3.25	3 19	3.13	3 24	3 17	3 41	32 22	32 93
16	3 19	3 25	3 21	3 14	3 30	3 19	3 28	3 25	3 46	3 08	32 33	32 96
17	3 46	3 32	3 23	3 23	3 24	3 13	3 09	3 23	3 31	3 25	32 49	33 09
18	3 39	3 25	3 18	3 29	3 20	3 23	3 39	3 15	3 17	3 18	32 43	33 10
	Total										549.74	561 29

i. Summation of individual cell voltages = 549.74 V
ii. Summation of cascade bus to bus voltages = 561.29 V
iii. Bus to bus voltage at entry to cell house = 566.5 V

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APPENDIX 14.1/3

MEASUREMENT OF INDIVIDUAL CELL VOLTAGES

CIRCUIT : X12

Date : 4.8.95

Time : 15.00 - 16.00 Hrs

Cascade No	B1	B2	B3	B4	B5	A5	A4	A3	A2	A1	Total	Bus to bus voltage
19	3 110	3 115	3 198	3 127	3 169	3 272	3 166	3 138	3 074	3 190	31 559	32 05
20	3 053	3 084	3 158	3 206	3 085	3 032	3 133	3 012	3 054	3 143	30 960	31 50
21	3 239	3 095	3 154	3 217	3 218	3 174	3 214	3 281	3 307	3 092	31 991	32 59
22	3 342	3 239	3 208	3 161	3 199	3 099	3.225	3.185	3.252	3.386	32 296	32 86
23	3 221	3 297	3 124	3 216	3 245	3 199	3 085	3 348	3 428	3 204	32 367	33 06
24	3 167	3 140	3 288	3 246	3 227	3 679	3 191	3 205	3 380	3 307	32 830	33 20
25	3 146	3 125	3 114	3 124	3 169	3.118	3 179	3 099	3 246	3 217	31 537	32.15
26	3 139	3 201	3 159	3 156	3 191	3 192	3.283	3.183	3 250	3 256	32 010	32 57
27	3 301	3 488	3 636	3 237	3 207	3 184	3 223	3 295	3 288	3 294	33 153	33 54
28	3 339	3 356	3 365	3 310	3 136	3 175	3 156	3 430	3 336	3 374	32 977	33 64
29	3 111	3 171	3 184	3 151	3 308	3.186	3 186	3.175	3 203	3 451	32 126	32 68
30	3 400	3 117	3 297	3 338	3 163	3 126	3 147	3 266	3 504	3 152	32 510	33 13
31	U N D E R M A I N T E N A N C E											-
32	3 103	3 112	3 185	3 329	3 204	3 166	3 263	3 163	3 245	3 316	32 086	32 60
33	3 199	3 213	3 405	3 144	3 191	3 263	3 170	3 173	3 370	3.193	32.321	32.82
34	3 247	3 235	3 167	3 173	3 244	3 361	3 357	3 362	3 220	3.362	32.728	33.36
35	3 160	3 343	3 261	3 329	3 155	3 280	3 363	3.342	3 235	3.372	32.840	33 36
36	3 188	3 181	3 262	3 252	3 186	3 160	3 231	3 219	3 140	3 256	32 075	32 66
	Total										548.37	557.77

i. Summation of individual cell voltages = 548.37 V
ii. Summation of cascade bus to bus voltages = 557.77 v
iii. Bus to bus voltage at entry to cell house = 560.90 V

APPENDIX 14.1/4

MEASURED MILLIVOLT DROPS ACROSS JUNCTIONS OF BUSBARS FROM RECTIFIER OUTPUT TO CELL HOUSE BUSBARS

Refer sketch 1 A enclosed.

mV drop across joints	X-12 Rectifier		X-22 Rectifier	
	+ ve Busbar	- ve Busbar	+ ve Busbar	- ve Busbar
A1	70 2	43 8	60	15 9
B1	8 2	3 8	42 1	79
C1	12 6	11 5	13 7	12 2
D1	8 5	8 7	4 52	15 2
E1	67 6	90 7	97 0	41 0
Main busbar joints below cell house (Nos 1-8)	12 8 to 30 mV		12 0 to 20 mV	
Average drop (mV)	20	20	20	20
No of joints	4	4	6	4
Total drop (mV)	80	80	120	80
Rectifier output (mV)	562 0 V	-	567 7 V	-
Output to cell house	560 0 V	-	565 4 V	-
mV drop in rectifier house	2 Volts	-	2 3 Volts	-

APPENDIX - 14.1/5

MEASURED MILLIVOLT DROPS ACROSS ANODIC AND CATHODIC JOINTS

A. CASCADE NO. : 5

i. Cell Reference : 5A2 Load : 12.5 kA

No.	Anode	Cathode	No.	Anode	Cathode	No.	Anode	Cathode
1	22.60	33.20	11	21.90	26.50	21	463.0	61.40
2	36.20	19.00	12	31.50	22.00	22	21.60	36.50
3	19.00	35.60	13	18.70	27.60	23	15.00	26.50
4	23.70	21.50	14	21.70	33.80	24	29.50	22.80
5	36.10	29.50	15	29.50	21.20	25	26.50	36.00
6	43.80	50.30	16	93.40	57.70	26	37.40	27.10
7	40.00	21.80	17	25.90	32.50	27	36.90	30.90
8	31.80	14.70	18	17.70	24.20	28	15.30	-
9	15.50	11.90	19	36.00	32.80	Avg.	29.8	31.90
10	20.00	46.00	20	31.20	23.70			

i. Average Millivolt drop across anode = 29.8 mV

ii. Average Millivolt drop across cathode = 31.9 mV

iii. Power loss across anode = (29.8 x 12.5)/1000
= 0.37 kW

iv. Power loss across cathode = (31.9 x 12.5)/1000
= 0.37 kW

v. Total loss = 0.37 + 0.37
= 0.74 kW

Appendix 14.1/5 contd..

ii. **CASCADE NO. : 5**

Cell Reference : 5A3 **Load : 12.5 kA**

No.	Anode	Cathode	No.	Anode	Cathode	No.	Anode	Cathode
1	21.19	60.50	11	24.50	55.50	21	31.90	72.40
2	22.50	52.10	12	55.90	40.00	22	21.00	32.80
3	36.40	37.00	13	28.80	80.90	23	24.80	49.00
4	29.60	48.50	14	23.11	73.40	24	36.90	44.00
5	81.80	50.50	15	81.70	60.40	25	27.50	48.00
6	27.00	35.20	16	28.90	40.50	26	29.80	50.50
7	26.30	43.80	17	80.40	81.50	27	45.00	137.2
8	35.30	38.30	18	28.60	38.00	28	15.00	-
9	40.11	26.90	19	43.00	64.80	Avg.	33.20	71.00
10	26.30	70.30	20	43.50	48.50			

i. Average Millivolt drop across anode = 33.2 mV

ii. Average Millivolt drop across cathode = 71 mV

iii. Power loss across anode = (33.2 x 12.5)/1000
= 0.42 kW

iv. Power loss across cathode = (71 x 12.5)/1000
= 0.89 kW

v. Total loss = 0.42 + 0.89
= 1.31 kW

Appendix 14.1/5 contd..

iii. CASCADE NO. : 5

Cell Reference : 5B3 Load : 12.5 kA

No.	Anode	Cathode	No.	Anode	Cathode	No.	Anode	Cathode
1	29.59	25.00	11	33.70	26.30	21	31.40	36.70
2	29.10	35.40	12	26.00	26.30	22	37.20	34.90
3	38.90	32.20	13	21.00	25.70	23	38.00	31.30
4	33.10	23.20	14	31.81	17.70	24	27.20	24.30
5	38.30	28.90	15	30.60	22.70	25	38.20	28.15
6	38.90	11.80	16	46.70	19.20	26	29.60	91.40
7	31.40	22.10	17	30.50	19.60	27	26.30	92.00
8	34.40	13.50	18	52.50	21.90	28	20.30	-
9	35.70	69.00	19	23.20	26.30	Avg.	43.40	32.80
10	29.50	44.20	20	28.00	36.00			

i. Average Millivolt drop across anode = 43.4 mV

ii. Average Millivolt drop across cathode = 32.8 V

iii. Power loss across anode = (43.4 x 12.5)/1000
= 0.54 kW

iv. Power loss across cathode = (32.8 x 12.5)/1000
= 0.41 kW

v. Total loss = 0.54 + 0.41

= 0.95 kW

Appendix 14.1/5 contd..

B. CASCADE NO. : 17

Cell Reference : 17A4 Load : 12.5 kA

No.	Anode	Cathode	No.	Anode	Cathode	No.	Anode	Cathode
1	26.20	88.40	11	45.70	66.20	21	36.70	38.10
2	46.10	42.00	12	49.70	45.50	22	25.60	26.10
3	31.10	50.00	13	25.30	33.50	23	40.50	36.50
4	52.70	39.00	14	25.31	44.00	24	20.50	41.60
5	52.50	42.40	15	24.40	34.60	25	25.00	32.40
6	31.60	31.60	16	28.60	32.00	26	19.80	31.70
7	22.00	53.30	17	48.00	62.30	27	34.00	52.90
8	30.60	186.0	18	37.50	70.60	28	23.10	-
9	58.40	183.5	19	15.30	27.80	Avg.	34.10	48.90
10	42.10	48.80	20	35.10	48.20			

i. Average Millivolt drop across anode = 34.1 mV

ii. Average Millivolt drop across cathode = 48.9 mV

iii. Power loss across anode = (34.1 x 12.5)/1000
= 0.43 kW

iv. Power loss across cathode = (48.9 x 12.5)/1000
= 0.61 kW

v. Total loss = 0.43 + 0.61
= 1.04 kW

Appendix 14.1/5 contd..

B. CASCADE NO. : 17

Cell Reference :17B5 Load : 13 kA

Date : 4.8.95

Time : 10.00 Hrs

No.	Anode	Cathode	No.	Anode	Cathode	No.	Anode	Cathode
1	23.10	27.00	11	30.50	19.80	21	31.50	37.60
2	24.10	28.90	12	27.10	18.30	22	51.10	30.60
3	49.00	21.70	13	58.10	21.90	23	56.50	26.10
4	31.20	19.80	14	45.70	24.50	24	42.50	40.00
5	26.50	21.50	15	48.30	25.30	25	51.10	19.50
6	32.70	21.50	16	42.90	12.60	26	22.90	24.30
7	33.60	17.70	17	41.40	18.10	27	25.90	27.00
8	39.60	70.60	18	22.90	27.50	28	19.44	-
9	34.90	21.70	19	20.60	28.30	Avg.	39.30	26.50
10	35.20	30.30	20	41.10	32.60			

i. Average Millivolt drop across anode = 39.3 mV

ii. Average Millivolt drop across cathode = 26.5 mV

iii. Power loss across anode = (39.3 x 13)/1000
= 0.51 kW

iv. Power loss across cathode = (26.5 x 13)/1000
= 0.34 kW

v. Total loss = 0.51 + 0.34
= 0.85 kW

Appendix 14.1/5 contd..

B. CASCADE NO. : 20

Cell Reference :20B5 Load : 13 kA

Date : 4.8.95

Time : 11.30 Hrs

No.	Anode	Cathode	No.	Anode	Cathode	No.	Anode	Cathode
1	50.30	53.01	11	37.10	38.20	21	42.80	25.30
2	36.50	33.30	12	29.00	26.20	22	36.60	33.00
3	41.40	95.10	13	22.20	47.00	23	30.50	18.10
4	36.90	71.10	14	11.00	73.50	24	43.10	26.20
5	46.50	58.40	15	22.00	11.70	25	31.20	14.00
6	54.10	32.90	16	37.30	40.00	26	33.10	29.50
7	38.10	44.60	17	34.50	90.00	27	22.90	27.10
8	49.30	48.00	18	35.60	40.00	28	18.50	-
9	36.50	40.00	19	48.30	42.00	Avg.	36.00	42.40
10	44.40	62.00	20	36.00	25.30			

i. Average Millivolt drop across anode = 36.0 mV

ii. Average Millivolt drop across cathode = 42.4 mV

iii. Power loss across anode = (36 x 13/1000)
= 0.47 kW

iv. Power loss across cathode = (42.4 x 13)/1000
= 0.55 kW

v. Total loss = 0.55 + 0.47
= 1.02 kW

Appendix 14.1/5 contd..

C. CASCADE NO. : 20

Cell Reference :20A5 Load : 13 kA

Date : 4.8.95

Time : 11.00 Hrs

No.	Anode	Cathode	No.	Anode	Cathode	No.	Anode	Cathode
1	30.40	63.90	11	36.80	12.50	21	22.30	73.90
2	38.70	27.80	12	47 00	9.700	22	15.60	-
3	35.60	30.20	13	31.00	31.50	23	20.00	53.20
4	38.20	43.90	14	48.30	26.10	24	40.30	35.00
5	35.80	27.50	15	17.60	71.50	25	20.90	-
6	19.60	22.90	16	33.10	59.00	26	15.00	-
7	49.00	38.50	17	38.40	22.60	27	-	-
8	32.80	27.10	18	52.10	28.90	28	-	-
9	30.00	11.80	19	15.60	23.80	Avg.	30.70	35.00
10	25.00	30.10	20	20.80	30.10			

i. Average Millivolt drop across anode = 30.7 mV

ii. Average Millivolt drop across cathode = 35 mV

iii. Power loss across anode = (30.7 x 13)/1000
= 0.41 kW

iv. Power loss across cathode = (35 x 13)/1000
= 0.46 kW

v. Total loss = 0.41 + 0.46
= 0.87 kW

Appendix 14.1/5 contd..

C. CASCADE NO. : 20

Cell Reference :20A2 Load : 13 kA

Date : 4.8.95

Time : 11.25 Hrs

No.	Anode	Cathode	No.	Anode	Cathode	No.	Anode	Cathode
1	6.900	19.50	11	38.90	44.50	21	24.40	17.00
2	15.80	16.50	12	79.30	35.80	22	42.20	34.80
3	13.50	22.90	13	50.40	22.04	23	95.00	31.20
4	15.80	21.20	14	69.20	114.0	24	45.00	35.50
5	7.300	21.60	15	47.80	71.50	25	52.00	25.30
6	22.20	16.60	16	33.10	24.90	26	53.00	18.20
7	24.60	20.90	17	24.80	12.50	27	26.30	-
8	22.00	18.80	18	25.70	9.550	28	-	-
9	39.90	15.00	19	15.90	38.75	Avg.	35.60	28.00
10	44.00	42.70	20	25.10	30.20			

i. Average Millivolt drop across anode = 35.6 mV

ii. Average Millivolt drop across cathode = 28 mV

iii. Power loss across anode = (35.6 x 13)/1000
= 0.46 kW

iv. Power loss across cathode = (28 x 13)/1000
= 0.36 kW

v. Total loss = 0.46 + 0.36
= 0.82 kW

Appendix 14.1/5 contd..

D. CASCADE NO. : 27

Cell Reference :27A2 Load : 13 kA

Date : 4.8.95

No.	Anode	Cathode	No.	Anode	Cathode	No.	Anode	Cathode
1	23.24	96.94	11	21.72	85.49	21	22.12	59.97
2	24.03	74.59	12	18.00	88.00	22	38.50	42.28
3	60.32	58.66	13	16.85	71.24	23	210.0	61.00
4	41.70	90.36	14	37.49	61.24	24	24.00	60.41
5	64.65	96.07	15	24.02	92.80	25	76.12	62.43
6	45.45	78.04	16	18.86	52.23	26	65.64	50.63
7	113.1	60.12	17	26.56	51.44	27	33.85	79.00
8	35.80	69.34	18	28.74	100.24	28	18.50	101.37
9	39.24	79.52	19	40.37	75.71	Avg.	45.61	70.24
10	75.45	55.29	20	32.82	82.27			

i. Average Millivolt drop across anode = 45.61 mV

ii. Average Millivolt drop across cathode = 70.24 mV

iii. Power loss across anode = (45.61 x 13)/1000
= 0.59 kW

iv. Power loss across cathode = (70.24 x 13)/1000
= 0.91 kW

v. Total loss = 0.59 + 0.91
= 1.5 kW

Appendix 14.1/5 contd..

D. CASCADE NO.: 27

Cell Reference :27A4 Load : 13 kA

Date : 4.8.95

No.	Anode	Cathode	No.	Anode	Cathode	No.	Anode	Cathode
1	8.94	55.26	11	40.47	60.27	21	43.35	94.95
2	13.32	62.72	12	26.07	55.34	22	72.75	95.98
3	39.82	64.10	13	29.42	19.66	23	33.75	-
4	34.48	74.33	14	13.94	78.84			
5	65.22	65.52	15	24.72	80.90			
6	24.30	61.28	16	14.20	50.41			
7	31.62	62.67	17	103.73	60.20			
8	17.40	55.10	18	43.99	110.73			
9	32.60	92.72	19	52.69	80.42	Avg.	37.65	69.52
10	25.56	77.36	20	73.68	71.24			

i. Average Millivolt drop across anode = 37.65 mV

ii. Average Millivolt drop across cathode = 69.52 mV

iii. Power loss across anode = (37.65 x 13)/1000
= 0.49 kW

iv. Power loss across cathode = (69.52 x 13)/1000
= 0.90 kW

v. Total loss = 0.49 + 0.90
= 13.9 kW

Appendix 14.1/5 contd..

D. CASCADE NO. : 27

Cell Reference :27B4 Load : 13 kA

Date : 4.8.95

No.	Anode	Cathode	No.	Anode	Cathode	No.	Anode	Cathode
1	26.74	25.73	11	54.09	33.70	21	22.46	81.50
2	173.0	48.00	12	17.72	19.33	22	21.86	54.00
3	29.81	37.17	13	67.52	73.81	23	28.05	55.76
4	15.50	62.48	14	24.70	36.10	24	29.26	29.42
5	37.90	55.51	15	38.76	17.63	25	40.64	56.48
6	40.90	77.63	16	39.82	29.45	26	24.56	
7	43.31	32.23	17	38.45	35.07	27	23.14	
8	35.43	42.43	18	31.32	28.63	28	18.40	
9	29.15	39.66	19	23.51	44.28	Avg.	38.85	43.01
10	73.18	33.97	20	38.59	25.31			

i. Average Millivolt drop across anode = 38.85 mV

ii. Average Millivolt drop across cathode = 43.01 mV

iii. Power loss across anode = (38.85 x 13)/1000
= 0.51 kW

iv. Power loss across cathode = (43.01 x 13)/1000
= 0.56 kW

v. Total loss = 0.51 + 0.56
= 1.07 kW

Appendix 14.1/5 contd..

D. CASCADE NO. : 27

Cell Reference :27B5 Load : 13 kA

Date : 4.8.95

No.	Anode	Cathode	No.	Anode	Cathode	No.	Anode	Cathode
1	19.40	39.57	11	41.01	51.04	21	64.69	61.69
2	14.40	31.90	12	38 03	70.65	22	25.65	40.21
3	19.45	27.65	13	40.33	57.03	23	16.43	44.86
4	61.20	32.57	14	40.33	40.46	24	25.90	39.13
5	50.51	36.57	15	30.27	33.33	25	49.49	56.34
6	28.52	22.13	16	22.91	29.87	26	47.47	56.68
7	7.98	41.09	17	22.30	38.87	27	34.87	33.95
8	57.63	45.42	18	38.34	67.12			
9	48.48	39.48	19	30.83	32.99	Avg.	35.38	44.84
10	111.34	63.59	20	67.58	76.43			

i. Average Millivolt drop across anode = 35.38 mV

ii. Average Millivolt drop across cathode = 44.84 mV

iii. Power loss across anode = (35.38 x 13)/1000
= 0.46 kW

iv. Power loss across cathode = (44.84 x 13)/1000
= 0.58 kW

v. Total loss = 0.46 + 0.58
= 1.04 kW

APPENDIX 14.1/6

QUANTIFICATION OF POWER LOSS DUE TO MILLIVOLT DROP ACROSS ANODIC AND CATHODIC JOINTS

A. Ist CIRCUIT : X22 RECTIFIER

Cell Reference	Average Millivolt drop		Power Loss		Total loss (kW)
	Anodic	Cathodic	Anodic (kW)	Cathodic (kW)	
5A2	29.8	31.9	0.37	0.37	0.74
5A3	33.2	71.0	0.42	0.89	1.31
5B3	43.4	32.8	0.54	0.41	0.95
Total			1.33	1.67	3.0

No. of cells per cascade = 10

Total power loss in cascade No.5 = 10/3 x 3

= 10 kW

Cell Reference	Average Millivolt drop		Power Loss		Total loss (kW)
	Anodic	Cathodic	Anodic (kW)	Cathodic (kW)	
17A4	34.1	48.9	0.43	0.61	1.04
17B5	39.3	26.5	26.5	0.34	0.85
Total			0.94	0.95	1.89

Total power loss in cascade No.17 = 1.89 x 10/2

= 9.45 kW

Total loss due to anodic and cathodic contact drops in Ist circuit = (10 +9.45) x 17/2

= 165 kW

Appendix 14.1/6 contd..

B. IInd CIRCUIT : X12 RECTIFIER

Cell Reference	Average Millivolt drop		Power Loss		Total loss (kW)
	Anodic	Cathodic	Anodic (kW)	Cathodic (kW)	
20B5	36.0	42.4	0.47	0.55	1.02
20A5	30.7	35.0	0.41	0.46	0.87
20A2	35.6	28.0	0.46	0.36	0.82
Total			1.34	1.37	2.71

Total power loss in the cascade = 2.71 x 10/3

= 9 kW

Cell Reference	Average Millivolt drop		Power Loss		Total loss (kW)
	Anodic	Cathodic	Anodic (kW)	Cathodic (kW)	
27A2	45.61	70.24	0.59	0.91	1.50
27A4	37.65	69.52	0.49	0.90	1.39
27B4	38.85	43.01	0.51	0.56	1.07
27B5	35.38	44.84	0.46	0.58	1.04
Total			2.05	2.95	5.00

Total power loss in the cascade = 5 x 10/4

= 12.5 kW

Total loss due to anodic and cathodic contact drops in IInd circuit = (9 + 12.5) x 17/2

= 183 kW

APPENDIX 14.1/7

MEASUREMENT OF MILLIVOLT DROP ACROSS BUSBAR JOINTS ON CELL TOP AND BOTTOM

X-22 RECTIFIER CIRCUIT

Cascade Ref.	Busbar Millivolt drop		Cascade Ref.	Busbar Millivolt drop	
	Cell top	Cell bottom		Cell top	Cell bottom
1A*	30	51.9	10A	8.26	31.15
1B*	35	51.9	10B	19.70	19.32
2A	17.32	96.30	11A	25.00	30.70
2B	19.95	73.76	11B	34.42	37.10
3A	14.02	30.92	12A	12.94	72.63
3B	12.98	27.32	12B	13.87	21.30
4A	7.05	62.0	13A	24.00	31.70
4B	11.35	27.34	13B	12.58	32.00
5A	9.00	32.30	14A	18.17	36.60
5B	15.05	43.40	14B	11.10	25.50
6A	73.18	47.00	15A	11.20	21.05
6B	21.10	15.20	15B	10.77	15.83
7A	28.52	15.90	16A	8.40	57.32
7B	17.80	51.0	16B	8.62	14.65
8A	7.35	48.70	17A	8.00	39.06
8B	16.50	43.20	17B	10.95	27.22
9A	9.69	50.30	18A	6.68	68.12
9B	8.75	15.30	18B	6.06	50.6
Total				593.42	1415.60

* Cell by-passed

Appendix- 14.1/7 contd..

MEASUREMENT OF MILLIVOLT DROPS ACROSS BUSBAR JOINTS ON CELL TOP AND CELL BOTTOM

X-12 RECTIFIER CIRCUIT

Cascade Ref.	Busbar Millivolt drop		Cascade Ref.	Busbar Millivolt drop	
	Cell top	Cell bottom		Cell top	Cell bottom
19A	6.30	59.50	28A	17.6	53.5
19B	5.50	67.30	28B	17.4	45.5
20A	10.7	51.20	29A	10.6	57.3
20B	9.7	46.6	29B	19.6	92.8
21A	25.7	92.5	30A	7.80	96.8
21B	19.7	41.1	30B	6.30	59.3
22A	7.30	48.9	31A	1.2*	31.7
22B	7.10	47.2	31B	10.4	39.1
23A	6.90	68.4	32A	9.4	31.4
23B	11.4	51.8	32B	9.8	71.5
24A	8.7	38.3	33A	7.2	57.8
24B	9.1	73.32	33B	11.4	35.2
25A	9.1	92.0	34A	16.9	21.7
25B	7.8	59.0	34B	18.1	9.00
26A	9.3	110.1	35A	14.9	18.0
26B	7.9	64.26	35B	6.4	20.0
27A	10.1	63.2	36A	14.3	20.0*
27B	14.2	72.6	36B	39.1	20.0@
Total				424.9 mV	1928 mV

* Cell by-passed
* Busbar is heavily pitted
@ Not measured due to space constraints of busbars

Appendix 14.1/7 contd..

X-22 CIRCUIT

Total milli volt drop across 'A' & 'B' side busbar joints on cell house floor	= 593.42 mV
Total milli volt drop across 'A' & 'B" side busbar joint on bottom cell	= 1415.60 mV

X-12 CIRCUIT

Total milli volt drop across 'A' & 'B' side busbar joints on cell house floor	= 424.9 mV
Total milli volt drop across 'A' & 'B" side busbar joint below cell house	= 1928 mV

Total power loss in circuit - II X-22/hr

$$= \frac{593.42 \times 13}{1000} + \frac{1415.6 \times 13}{1000}$$

= 7.7 + 18.4

= 26.1 kW

Total loss in circuit - I ie., X-12/hr

$$= \frac{424.9 \times 13}{1000} + \frac{1928 \times 13}{10000}$$

= 5.523 + 25.064

= 30.587 kW

APPENDIX 14.1/8

POWER LOSS DUE TO RESISTANCES OF ANODE AND CATHODE ELECTRODES

Sl. No.	Parameter	Anode	Cathode
i	Resistivity of lead	20.8 Micro ohm cm (0.0000208 ohm cm)	3.21 Micro ohm cm (0.00000321 ohm cm)
ii	Area cm^2 (m^2)	40.6 (0.00406)	30.5 (0.00305)
iii	Dimensions Length = cm Width = cm Thickness = cm	 113 58 0.7	 115 61 0.5
iv	Resistance (Micro ohms)	ρl/a 57.89	ρl/a 12.1
v	Load kA (E)	13000	13000
vi	Loss in electrode (kW/cell)	$\frac{I^2 \times (iv)}{1000}$ = 9.8	$\frac{I^2 \times (iv)}{1000}$ 2.0

APPENDIX 14.1/9

POWER LOSS DUE TO ELECTROLYTE RESISTANCE IN THE CASCADE

i. Total Resistance in cell

$R = R_K + R_A + R_E$

R_K = Resistance of cathode
R_A = Resistance of anode
R_E = Resistance of electrolyte

ii. Average applied voltage/cell = 3.1 V

iii. Theoretical decomposition voltage (for 65 mm spacing) = 2.68 V

iv. Driving voltage = 0.42 V

v. Current flowing from anode to cathode $= \frac{13000}{28 \times 2}$

= 232 Amps

vi. Resistance of each current path = 0.42/232

= 1.81 Milli ohms

= 1810 Micro ohms

vii. Combined resistance of electrolyte path per cell for 28 paths $= \frac{0.42}{232 \times 28}$

= 0.0646 Milli Ohms

viii. Power loss due to electrolyte per cell $= I^2R$

$= \frac{13000 \times 13000 \times 0.0646}{1000 \times 1000}$

= 10.9 kW

ix Power loss due to electrolyte per cascade = 109 kW

Appendix 14.1/9 contd..

ESTIMATION OF TOTAL LOSSES FOR A TYPICAL CASCADE

Sl. No.	Loss Area	Actual loss (kW)	% Loss
1.	ii. Busbar contacts cell top iii. Busbar contacts cell bottom	7.7 18.4 ----- 26.1 -----	16.7
2.	i. Anodic contact drops ii. Cathodic contact drops	4.4 5.6 ----- 10.0 -----	6.3
3.	ii. Resistance due to anode electrode iii. Resistance due to cathode electrode	9.8 2.0 ---- 11.0 -----	7.0
4.	Loss in electrolyte	109	70
	Total	156.1	100

APPENDIX 14.1/10

OBSERVATIONS ON FEED AND SPENT ELECTROLYTE OF SELECTED CELLS IN CELL HOUSE

Date : 5.8.95

i. Avg. Feed electrolyte composition :

Zn - 50.00 gpl, Acid - 142 gpl, Co - 0.3 mgp, Ni < 0.3 mspl, Mn - 2.3 gpl

ii. Avg. Spent electrolyte composition

Zn - 43.8 gpl, Acid - 151 gpl, Mn - 2.2 gpl

Sl. No.	Parameter	Time (Hrs)		
		9.30	10.30	11.30
1.	Feed electrolyte acidity (g/l)	135.0	136.0	137.0
2.	Spent electrolyte acidity (g/l)	145.0	146.0	147.0
3.	Load of X-12 rectifier circuit in kA	12.0	12.0	13.0
4.	Load of X-22 rectifier circuit in kA	11.0	11.0	13.0

AVERAGE INLET AND OUTLET TEMPERATURES OF SELECTED CELLS

Date : 5.8.95

Time : 9.30 A.M. to 12.00 Noon

Sl. No.	Cell No.	Side	Inlet temp. °C	Outlet temp. °C	Temp. rise (ΔT) °C	Prodn. (MT)
1	5	A	37.33	44.46	+ 7.13	1.490
		B	37.67	42.10	+ 4.43	1.560
2	17	A	37.60	47.47	+ 9.87	1.550
		B	37.80	44.40	+ 6.60	1.430
3	20	A	37.06	47.60	+ 10.54	1.300
		B	37.20	45.53	+ 8.33	1.510

APPENDIX 14.1/11

OBSERVATIONS ON SPENT ELECTROLYTE COOLERS

Sl No	Parameter	Cooler 1	Cooler 2	Cooler 3	Cooler 4	Cooler 5	Cooler 6
1.	Average inlet temp °C	41	-	41	41	41	41
2	Average outlet temp from coolers (°C)	35 60	-	34 00	34 80	35 00	35.60
3	Design drop	6 0	-	6 0	6 0	6 0	6 0
4	Temperature drop across cooler (°C)	5 4	-	7 0	6 2	6 0	5 4
5	Operation of fan	Y	N	Y	Y	Y	Y
6	No of belts (Nos)	3	-	2	3	2	2

APPENDIX - 14.1/12

ELECTROLYSIS PLANT - BREAK-UP OF POWER

Power Parameters	Power in kW	Power in %
Average power input	7315	100
Rectifier transformer losses	273.8	3.74
Losses in DC distribution	29.3	0.4
Inter cell anode/cathode loss	165	2.25
Loss in electrolyte	1853	25.33
Power for electrolysis	4993.2	68.26

OBSERVATIONS ON ZINC MELTING FURNACES

A. OBSERVATIONS ON AJAX INDUCTION FURNACE

One scoop 5 castings

Set point - 530 °C

Time	AJAX temp. °C	Line Current			PF	V
250 PM	450	680	415	515	0.96	560
300 PM	460	660	410	500	0.96	540
310 PM	465	660	410	500	0.96	550
325 PM	470	660	410	500	0.965	550
330 PM	460	670	410	500	0.96	550
340 PM	460	670	415	510	0.96	555
350 PM	460	660	415	510	0.96	555
400 PM	470	660	415	500	0.97	540
410 PM	475	650	410	500	0.965	540
420 PM	475	650	410	500	0.965	540
430 PM	470	675	410	505	0.96	550
	Avg.465	Avg.335	-	-	-	-

i. Quantity of material charged = 8400 kgs

ii. Observation time = 1.75 Hrs

iii. Average power consumption = 580 kW

iv. Hourly charging of material = 8400/1.75
= 4800 kgs/hr

v. Specific power consumption = 580/4.8
= 121 kWh/MT

Appendix 14.2/1 contd..

B. OBSERVATIONS ON RUSSIAN INDUCTION FURNACE

Date : 7.8.95

MELTING BATH SET TEMPERATURE : 500 °C

Sl. No.	Time (Hrs)	Front bath °C	Melting Bath °C	Voltage		Line Current Heater						Cos ϕ
				I V	II V	I A	II A	III A	IV A	V A	VI A	
1	9.00	-	-	380	360	260	230	240	250	250	260	0.9
2	9.20	-	470	380	360	260	230	240	250	250	270	0.9
3	9.40	-	470	370	360	260	230	240	250	250	260	0.9
4	10.05	-	470	370	360	260	230	240	245	250	265	0.9
5	10.35	-	470	370	360	260	230	240	250	250	260	0.9
6	11.05	-	470	380	360	260	230	245	250	250	260	0.9
7	11.30	-	460	380	360	260	230	240	250	250	260	0.9
8	12.00	-	460	380	360	255	230	240	245	250	275	0.9
9	15.00	-	490	370	360	260	230	245	245	250	280	0.9
10	16.25	-	480	380	370	270	240	250	255	260	290	0.9
			Avg.471									

MATERIAL CHARGED

Sl. No.	Time (Hrs)	kgs
1	10.00 - 11.00	3320
2	11.00 - 12.00	4590
3	12.00 - 14.00	4390
4	14.00 - 15.00	2960
5	15.00 - 16.00	2900
Total		18160

i. Power input = 1.732 x 0.370 x 490 x 0.9
= 283 kW

ii. Quantity of material charged = 18,160 kgs

iii. Duration = 6 Hrs

iv. Hourly material charging rate = $\frac{18,160}{6}$

= 2,018 kgs

v. Specific power consumption = $\frac{283}{2.018}$

= 140 kWh/MT

APPENDIX - 14.2/2

THEORETICAL POWER REQUIREMENT FOR ZINC MELTING

Melting point of Zinc	= 419.5 °C
Heat of fusion of Zinc	= 1595 cal/mole
Specific heat of Zinc	= 0.0918 & 0.118 cal/gm °C
No. of moles/ton	= 1000000/65.38
	= 15295.2 gm mole
Heat required to melt 1 ton of zinc	= 1000000 x 0.0915 x(419.5-30) + 15295.2 x 1595 + 0.118 x (465 - 419.5) x 1000000
	= 65404094/1000/860
	= 76.0 kWh/ton of zinc
Existing power consumption for 1 ton of zinc	= 121 kWh/MT to 140 kWh/MT
Therefore efficiency of melting	= 54 % to 63 %

APPENDIX - 14.2/3

SURFACE HEAT LOSSES FROM RUSSIAN AND AJAX FURNACES

RUSSIAN FURNACE

Date :7.8.95

Ambient air temperature : 35 °C

Surface heat loss from a surface is estimated by using the expression

$$= \left[10 + \frac{(T_s - T_a)}{20} \right] (T_s - T_a) \text{ .... kcal/hr/m}^2$$

Where T_s - Surface temperature (°C)

T_a - Ambient air temperature (°C)

Sl. No.	Area/Section	Average Surface temp. (°C)	Heat loss kcal/hr/m^2	Area m^2	Surface heat loss kcal/hr
1.	Left (i)	62	306.45	2.445	749.27
	(ii)	74	466.05	2.445	1139.49
	(iii)	82	380.45	2.445	1419.20
2.	Back	58	256.45	1.750	448.78
		79	536.80	1.750	939.57
3.	Right	60	281.25	3.36	945.00
		76	494.05	3.36	1660.0
4.	Top	57	244.20	7.70	1880.34
	Total			25.255	9181.65

*** Areas facing material removal**

SUMMARY OF HEAT LOSS

Sl. No.	Area/Section	Surface heat loss (kcal/hr)	Area (m^2)	% of Total heat loss	% of Total area
1.	Left	3307.96	7.335	36.03	29.04
2.	Back	1388.35	3.500	15.12	13.86
3.	Right	2605.00	6.72	28.37	26.61
4.	Top	1880.34	7.70	20.48	30.49

Appendix 14.2/3 contd..

AJAX FURNACE

Date : 7.8.95

Ambient air temperature = 35 °C

Sl. No.	Area/Section	Average Surface temp. (°C)	Heat loss kcal/hr/m^2	Area m^2	Surface heat loss kcal/hr
1.	Blower side	51	172.80	13.14	2270.60
2.	Inductor side	52 72	184.45 438.45	6.57 6.57	1211.84 2880.61
3.	Backside	55	220.0	2.86	629.20
Total				29.14	6992.25

SUMMARY OF HEAT LOSS

Sl. No.	Area/Section	Surface heat loss (kcal/hr)	Area (m^2)	% of Total heat loss	% of Total area
1.	Blower side	2270.60	13.14	32.47	45.10
2.	Inductor side	4092.45	13.14	58.53	45.10
3.	Back side	629.20	2.86	8.997	9.80
Total		6992.25	29.14	99.99	100.00

APPENDIX - 15/1

LEAD SMELTING

PROCESS FLOW CHART

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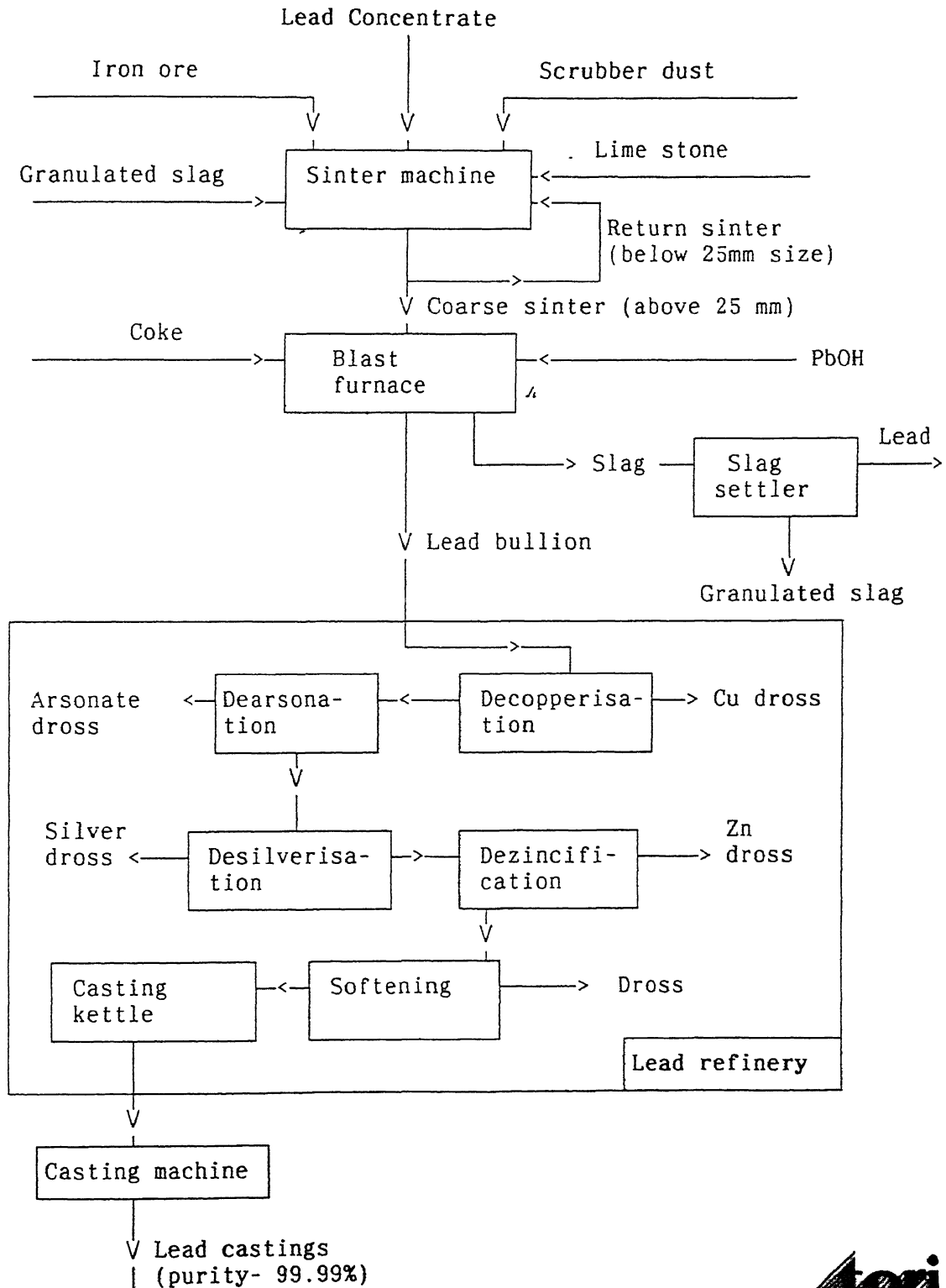
APPENDIX - 15.1/

MONTH-WISE ENERGY CONSUMPTION AND PRODUCTION IN SINTER MACHINE FOR THE YEAR 1994-95

Month	LDO kL	Production MT	Operating hrs	Specific energy consumption kL/MT
Apr 94	32	3388	407:10	9.445
May	40	3144	408:25	12.723
Jun	20	1798	273:50	11.123
Jul	20	4104	544:25	4.873
Aug	15	2985	339:30	5.025
Sep	25	3736	473	6.692
Oct	25	3772	462:15	6.628
Nov	30	3321	434:10	9.033
Dec	36	3362	445:10	10.708
Jan 95	15	1850	227:50	8.108
Feb	20	2806	357:15	7.128
Mar	30	3614	509:35	8.301
Total	308	37880	-	8.316

Min. Specific energy consumption = 4.873 L of LDO/MT

Max. Specific energy consumption = 12.723 L of LDO/MT

Avg. Specific energy consumption = 8.316 L of LDO/MT

APPENDIX - 15.1/2

INPUT MATERIALS TO THE FURNACE

1. Through Feeders

Feed No.	Material	Input rate MT/hr
2	Return sinter	18.00
4	Slag	0.70
5	Coke Breeze	0.40
6	Lime stone	0.45
7	Iron ore	0.50
8	Lead concentrate (scrubber dust)	6.50
	Total	26.55

2. Air through Blowers

Sl No.	Blower	Air in m^3/min
1.	Combustion air	22.5
2.	Fresh air	172.0
	Total	194.50

Weight of air = 194.5 x 1.21

= 235.34 kg/min

= 14120 kg/hr

= 14.12 MT/hr

3. Water through drum mixers

Sl No.	Drum Mixer	Water qty. l/hr
1.	Near charge preparation	500.0
2.	Near sinter furnace	500.0
	Total	1000.00

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Appendix - 15.1/2 contd..

Total water input = 1 MT

4. Fuel input = 58.85 lt of LDO/hr

= 58.85 x 0.85

= 50 kg/hr

= 0.05 MT/hr

Total input material = Feeder material + air + water + fuel (1 + 2 + 3 + 4)

= 26.55 + 14.12 + 1 + 0.05

= 41.72 MT/hr

APPENDIX - 15.1/3

OBSERVED PARAMETERS IN SINTERING FURNACE

Particulars	Time (Hours)				
	11.15	11.45	12.15	14.00	15.00
FEEDER CONTROL POSITION					
Feeder 1 - Return sinter	-	-	-	-	-
Feeder 1 - Return sinter	180	180	190	230	230
Feeder 3 - Spare	-	-	-	-	-
Feeder 4 - Slag	70	70	70	70	70
Feeder 5 - Coke Breeze	500	430	430	500	550
Feeder 6 - Lime Stone	80	50	50	50	50
Feeder 7 - Iron ore	300	300	300	290	290
Feeder 8 - Lead Concentrate	370	370	370	370	370
TEMPERATURES °C					
Sinter hood 1	200	200	240	165	175
Sinter hood 2	300	320	400	270	250
Sinter hood 3	290	350	340	250	270
Gas temp	208	200	250	166	155
Recirculation duct	210	270	250	200	210
AIR FLOWS m^3/min					
Fresh air	150	175	185	178	170
Recirculation air	138	150	170	175	110
WIND BOX PRESSURES (X6.3MM WG)					
Fresh air					
Hood 1	-	-	-	-	-
Hood 2	40	46	40	40	33
Hood 3	-	-	-	-	-

Appendix - 15.1/3 contd..

Sulphide sulphur		= 6-7%
Wind Box pressures		
Fresh air pressure	Hood 1	= 250-325 mmwg
	Hood 2	250-325 mmwg
	Hood 3	250-325 mmwg
Recirculated air	Hood 1	= 225-325 mmwg
Wind box pressure	Hood 2	= 225-325 mmwg
Sinter hood temp.	Hood 1	= 160-200°C
	Hood 2	250-350°C
	Hood 3	230-330°C
Sinter hard pressure	Hood 1	= 0.3-0.6 mmwg
	Hood 2	1.5-2.0 mmwg
	Hood 3	1.0-2.0 mmwg
Recirculation gas temp.		= 200-250°C

Parameter to be maintained at gas cleaning section

Hot gas blower suction	= 800-1000 mmwg
Humidifier inlet gas temp.	= 150-250°C
Humidifier outlet gas temp.	= 50-60°C
Hot gas blower discharge pr.	= 800-300 mmwg
Pressure drop across tray separator	= 250-300 mmwg
Pressure dross across ventury	= 500-650 mmwg
Pressure drop across precipitator	= 20 mmwg
50 TPD blower suction	= 0-100 mmwg

Sintering furnace reactions

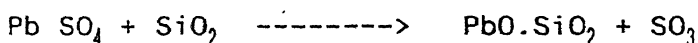
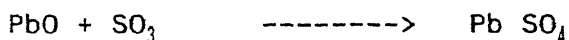
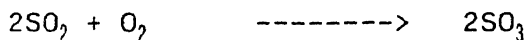
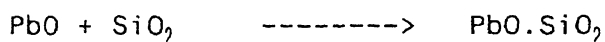
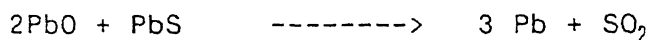
$2PbS + 3\ O_2$ ---------> $2\ PbO + 2SO_2$ + 99.9 kcal

$2PbO + PbS$ ---------> $3\ Pb + SO_2$

$PbO + SiO_2$ ---------> $PbO.SiO_2$

$2SO_2 + O_2$ ---------> $2SO_3$

$PbO + SO_3$ ---------> $Pb\ SO_4$

$Pb\ SO_4 + SiO_2$ ---------> $PbO.SiO_2 + SO_3$

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APPENDIX - 15.1/4

HEAT BALANCE OF SINTER MACHINE

1. Heat Inputs

   a. Heat given through fuel
   b. Heat given through coke breeze
   c. Heat given by exothermic reaction

2. Heat Outputs

   a. Heat given to the material
   b. Heat loss due to recirculation of air
   c. Heat loss due to H_2O in feed material
   d. Heat loss due to H_2O in air
   e. Heat loss due to surface heat losses
   f. Heat loss due to vertical and horizontal jacket cooling by water
   g. Heat loss due to recirculation of ignition air
   h. Heat given to exhaust gases
   i. Calcination of lime stone
   j. Unaccounted losses

1. Heat Inputs

a. Heat given through fuel

LDO consumption = 50 kg/hr

Heat given through LDO = 50 x 10800

= 540000 kcal/hr

b. Heat given through coke breeze

Coke breeze consumption = 400 kg/hr

Heat given through coke breeze = 400 x 5500

= 2200000 kcal/hr

c. Heat given by exothermic reaction

Total exhaust gases = 10000 Nm^3/hr

Percentage of SO_2 in exhaust gases = 1.5%

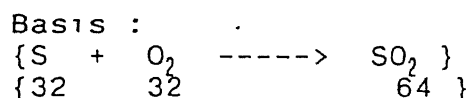
Quantity of SO_2 = 10000 x 0.015

= 150 Nm^3/hr

= 150 x 2.93 kg/hr

= 439.5 kg/hr

Quantity of sulphur reacted to form SO_2 :

Basis :
{S + O_2 -----> SO_2 }
{32   32   64 }

1 kg of S + 1 kg of O_2 ---> 2 kg of SO_2

Qty of sulphur reacted = 439.5 x 0.5

= 219.75 kg/hr

Heat given by sulphur = 219.75 x 2200

= 483450 kcal/hr

Total heat input = (a + b + c)
= 540000 + 2200000 + 483450

= 3223450 kcal/hr

I. Heat Outputs

a. Heat given to the material

i. Heat given to sinter returns and scrubber dust
ii. Heat given to lime stone
iii. Heat given to slag
iv. Heat given to iron ore

i. Heat given to sinter returns and scrubber dust

Heat given to lead and associate material

Total lead and associate material composition = 53%

Total return sinters and scrubber dust = 18 + 6.5

= 24.5 MT/hr

Lead and associate material = 24.5 x 0.53 .

= 12.985 MT/hr

= 12985 kg/hr

Material outlet temperature = 550°C

Heat given to lead and associate material = 0.03x (550-30) x 12985

= 202566 kcal/hr

Gangue material in the feed material = 47%

Total gangue material = 24.5 x 0.47

= 11.515 MT/hr

= 11515 kg/hr

Heat given to gangue material = 0.13 x 11515 (550-30)

= 778414 kcal/hr

Total heat given to return sinter material & scrubber dust and gangue material = 202566 + 778414

= 980980 kcal/hr

ii. **Heat given to lime stone**

Total $CaCo_3$ feed rate = 400 kg/hr

Output material due to calcination $CaCo_3$ converted to CaO & CO) = Cao

Quantity of CaO (1 kg of $CaCO_3$ gives 0.56 kg of Cao) = 0.56 x 400

= 224 kg/hr

Heat given to lime (Cao) = 0.22 x 224 (550-30)

= 25625 kcal/hr

iii. **Heat given to slag**

Total slag feed rate = 700 kg/hr

Heat given to slag = 0.30 x 700 (550-30)

= 109200 kcal/hr

iv. **Heat given to Iron ore**

Iron ore feed rate = 500 kg/hr

Heat given to iron ore = 0.13 x 500 (550-30)

= 33800

Total heat given to material (i + ii + iii + iv) = 980980 + 25625 + 109200 + 33800

= 1149605 kcal/hr

Appendix - 15.1/4 contd..

b. Heat loss due to recirculation of air

Exit temp. of recirculation air	= 271°C
Inlet temp. of recirculation air	= 131°C
Quantum of recirculating air	= 8772 m^3/hr
Weight of recirculation air	= 8772 x 1.21
	= 10614 kg/hr
Heat loss due to recirculation of air	= 0.21 x 10614 (271-31)
	= 312051 kcal/hr

c. Heat loss due to H_2O in feed material

Total quantity of H_2O in feed	= 1000 kg/hr
Inlet water temperature	= 30°C
Exhaust gas temperature	= 200°C
Heat given to water to form vapour	= 1000{(100-30) + 540+0.5 (200-100)}
	= 6600000 kcal/hr

d. Heat loss due to H_2O in air

Dry bulb temperature	= 30°C
Wet bulb temperature	= 26°C
Total air supplied (fresh air + combustion air)	= 194.5 Nm^3/min
	= 14120 kg/hr
Specific humidity in air	= 0.02 kg/kg of air
Total water vapour in the air	= 282.4 kg/hr
Heat given to the vapour	= 282.4 x 0.5 (200-30)
	= 24004 kcal/hr

Appendix - 15.1/4 contd..

e. Heat loss due to surface heat losses

Surface heat losses :

Sl No	Particulars	Area m^2	Temp °C	Rad Loss kCal/hr	Con Loss kCal/hr	Tot loss kCal/hr	kCal/hr per m^2
1.	**Sinter Machine**						
	Left hand side	16 38	130	10751 47	8775 64	19527 10	1192 13
	Right Hand side	16 38	120	9240 57	7692 75	16933 32	1033 78
	Top	45 74	130	30022 71	31844 51	61867 22	1352 58
	Total (1)	78 50		50014 75	48312 90	98327 65	1252 58
	Material outlet chamber						
	Left hand side	5 20	120	2933 51	2442 14	5375 66	1033 78
	Right Hand side	5 20	105	2279 58	1944 44	4224 02	812 31
	Top	7 20	140	5441 37	5646 92	11088 29	1540 04
	Total (2)	17.60		10654 46	10033 51	20687 97	1175 45
	Grand Total (1+2)	96 10		60669 21	58346 40	119015 62	1238 46

Total surface heat losses = 119015 kcal/hr

f. Heat loss due to vertical and horizontal jacket cooling by water in ignition chamber

Water flow rate in hood :
Vertical = 1000 kg/hr
Horizontal = 1000 kg/hr

Inlet water temp. to the jackets = 43 °C

Outlet water temp.
Vertical jacket = 46°C
Horizontal jacket = 50°C

Heat given to cooling water
Vertical jacket = 1000 (46 - 43)
= 3000 kcal/hr

Horizontal jacket = 1000 (50- 43)
= 7000 kcal/hr

Total heat given to cooling water in both jackets = 10000 kcal/hr

Appendix - 15.1/4 contd..

g. Heat loss due to recirculation of ignition air

Quantity of ignition air (recirculation) = 2500 m^3/hr

= 3000 kg/hr

Outlet temp. of ignition air = 139°C

Inlet temp. of ignition air = 100°C

Heat loss due to recirculation = 3000 x 0.21 (139-100)

= 24570 kcal/hr

h. Heat given to exhaust gases

Exhaust gas flow rate = 10000 Nm^3/hr

Exhaust gas analysis
CO_2 = 8.5%
O_2 = 1.5%
SO_2 = 1.5%
N_2 = 88.5%

Exhaust gas temp. = 200°C

Exhaust gas losses :

Element	% V/V	Density kg/m^3	Vol. m^3/hr	Sp.heat kcal/m^3.c	Heat loss kcal/hr
CO_2	8.5	1.977	850	0.40	57800
O_2	1.5	1.43	150	0.30	7650
SO_2	1.5	2.92	150	0.43	10965
N_2	88.5	1.250	8850	0.31	466395
Total			100000		542810

i. Heat given for calcination of lime stone

Total $CaCO_3$ supply rate = 450 kg/hr

1 kg of $CaCO_3$ forms = 0.56 kg of Cao

Total Cao formation = 0.56 x 450
= 252 kg/hr

Heat of reaction (endothermic) = 1 kg of $CaCo_3$ requires 535 kcal

= 535 x 450

= 240750 kcal/hr

Appendix - 15.1/4 contd..

j. **Unaccounted losses**

= 140645 kcal/hr

HEAT BALANCE SHEET

Particulars	kcal/hr	Percentage
Heat Input		
Heat given through fuel	540000	16.75
Heat given through coke breeze	2200000	68.25
Heat given by exothermic reaction	483450	15.00
Total	3223450	100.00
Heat Output		
Heat given to the material	1149605	35.66
Heat loss due to recirculation of air	312051	9.68
Heat loss due to H_2O in feed material	660000	20.47
Heat loss due to H_2O in air	24004	0.74
Heat loss due to surface heat losses	119015	3.69
Heat loss due to vertical & horizontal jacket cooling by water	10000	0.76
Heat loss due to recirculation of ignition air	24570	0.31
Heat given to exhaust gases	542810	16.84
Heat given to calcination of lime stone	240750	7.48
Unaccounted losses	140645	4.37
Total	3223450	100.00

APPENDIX - 15.1/5

UTILISATION OF HEAT IN RECIRCULATION AIR FOR PREHEATING COMBUSTION AIR

The air in the sintering furnace is being recirculated through a blower, to cool the furnace and feed material and enrich the air for SO_2.

A. Data

i. Recirculation air flow rate = 8772 m^3/hr

= 8772 x 1.21

= 10614 kg/hr

ii. Temp. of recirculation air

Outlet of furnace = 271°C
Inlet to furnace = 131°C

iii. Heat loss in recirculation = 0.21 x 10614 (271-131)

= 312051 kcal/hr

B. Heat Recovery

The heat loss in recirculation can be recovered by preheating the combustion air upto 200°C.

i. Combustion air flow rate = 1350 m^3/hr

= 1350 x 1.2

= 1620 kg/hr

ii. Combustion air inlet temp. = 30°C

iii. Recoverable heat by preheating air upto 200°C = 0.21 x 1620 (200-30)

= 57834 kcal/hr

iv. Savings in LDO $= \frac{57834}{10800}$

= 5.355 kg/hr

Appendix - 15.1/5 contd..

v. Annual savings of LDO @ 6000 operating hrs/yr = 5.355 x 6000

= 32130 kg/year

= 32.130 MT/year

= 32130/0.85

= 37.8 kL/year

vi. Cost savings = 37.80 x 7310

= Rs.2.763 lakhs/year

C. Required Heat Transfer Area

i. LMTD

$$LMTD = \frac{HTD - LTD}{\log_e HTD/LTD}$$

where, HTD = higher temperature difference
= 221°C
LTD = Lower temperature difference
= 101°C

$$LMTD = \frac{101 - 70}{\log_e 101/70}$$

= 84.55

ii. Required heat transfer area

a. Recoverable heat = 57834 kcal/hr

= UA (LMTD)

b. Overall heat transfer co-efficient = 10 kcal/hr m^2°C

c. Heat transfer area $= \frac{57834}{84.55 \times 10}$

= 68 m^2

D. Investment required = Rs.6 lakhs

E. Payback period = 2.17 years

APPENDIX - 15.1/6

USE OF FURNACE OIL IN SINTER MACHINE

A. Data

i.	LDO consumption (Avg)	= 50 kg/hr
		= 58.83 lts/hr
ii.	Hourly cost of LDO	= 58.83 x 7.31
		= Rs.430/hour
iii.	Calorific value of LDO	= 10800 kcal/kg
iv.	Specific energy cost of LDO	$= \frac{10800 \times 0.85}{7.31}$
v.	Hourly heat requirement	= 1255.81 kcal/Re
		= 50 x 10800
		= 540000 kcal/hr

B. Analysis and Recommendation

Use of FO in sinter machine will result in cost savings

i.	FO calorific value	= 10200 kcal/kg
ii.	FO oil requirement/hr	$= \frac{540000}{10200}$
		= 52.94 kg/hr
		= 52.94/0.95
		= 55.75 lts/hr
iii.	Hourly cost of FO burning	= 55.75 x 5.344
		= Rs.298.00
iv.	Power required for heating of FO	= 0.07 kW/L
v.	Total power requirement	= 3.5 kW
vi.	Cost of heating Rs./hr	= 3.5 x 2.59
		= Rs.9.00/hr

Appendix - 15.1/6 contd..

vii. Total cost of FO heating = Fuel cost + heating cost

= Rs.298 + 9

= Rs.307/hour

Specific cost of energy kcal/Re = $\frac{10200 \times 52.94}{298}$

= 1812 kcal/Re.

C. Savings

Hourly cost savings = Rs. (430 - 307)

= Rs.123/-

Annual cost savings = Rs.123 x 300 x 24

= Rs.8.856 lakhs

Equivalent LDO savings kL/year = $\frac{885600}{7310}$

= 121.14 kL/year

D. Investment required = Nil

E. Simple payback period = Immediate

APPENDIX - 15.1/7

COMBINED EFFICIENCY EVALUATION OF FRESH AIR AND RECIRCULATION BLOWER IN SINTERING FURNACE

1. Fresh Air Blower

i. Fan air flow rate = 172 m^3/min

= 10320 m^3/hr

ii. Fan air pressure = 600 mm wg

iii. Rated flow = 12000 m^3/hr

iv. Actual power consumption = 29.01 kW

v. Theoretical power reqd. $= \frac{m^3/min \times mmwg}{6120}$

$= \frac{172 \times 600}{6120}$

= 16.86 kW

vi. Actual power consumption = 29.01

vii. Efficiency of fan (Fan and Motor) $= \frac{\text{Theoretical power}}{\text{Actual power}} \times 100$

$= \frac{16.86}{29.01} \times 100$

= 58.11%

viii Percentage fan output $= \frac{10320}{12000} \times 100$

= 86%

Appendix - 15.1/7 contd..

2. Recirculation Fan

i. Fan air flow rate = 146.2 Nm^3/hr

= 8772 m^3/hr

ii. Design air flow rate = 12000 Nm^3/hr

iii. Percentage output = $\frac{8772}{12000}$

= 73.1%

Fan	Output m^3/min	Percentage output	Efficiency
Fresh air	172.0	86.0	58.11
Recirculating fan	146.2	73.1	-

APPENDIX - 15.2/1

MONTH-WISE ENERGY CONSUMPTION AND PRODUCTION IN BLAST FURNACE FOR THE YEAR 1994-95

Month	Coke MT	Production MT	Operating hrs	Specific energy consn. kg of Coke/MT
Apr 94	570.00	1202.00	548:30	474.210
May	520.00	1060.00	533:35	490.566
Jun	250.00	508.00	393:45	492.126
Jul	740.00	1703.00	59:05	434.527
Aug	240.00	1002.00	415:05	239.521
Sep	190.00	1128.00	532:05	168.440
Oct	336.59	1054.00	563	319.345
Nov	480.00	926.00	476:35	518.359
Dec	500.00	1050.00	535:50	476.190
Jan 95	233.00	380.00	198:25	613.158
Feb	380.00	810.00	388:40	469.136
Mar	635.00	1220.00	497:25	520.492
Total	5074.59	12043.00		434.672

Min. Specific energy consumption = 168.440 kg of coke/MT

Max. Specific energy consumption = 613.158 kg of coke/MT

Avg. Specific energy consumption = 434.672 kg of coke/MT

APPENDIX - 15.2/2

DESIGN OPERATING PARAMETERS OF BLAST FURNACE

Structural details

Size	= 1.9 m x 1.3 m at top
Height	= 5.38 m
No.of tuyers	= 30
Tuyers area	= 4.68 m^2
Tuyers dia	= 80 mm/50 mm
No.of rows of tuyers	= One
Crucible depth	= 500 mm
Length of water jackets	= 1.7 m

Operational details

Blast air volume	= 6000-6200 Nm^3/hr
Pressure	= 1100-1400 mmwg
Slag temp.	= 1100-1200°C
Lead temp.	= 800-1000°C
Oxygen flow	= 90-120 m^3/hr
Oxygen enrichment	= 1-1.5%

Charge composition

Sinter	= 4000 kgs
Coke	= 750-800 kgs

Slag composition

Pb	= 2-2.5%
SiO_2	= 20-25%
CaO	= 14-16
FeO	= 34-35
ZnO	= 6-10
Al_2O_3	= 6-8
Ag	= 20-60 gPJ
Basicity	= 0.8-8.5

Appendix - 15.2/2 contd..

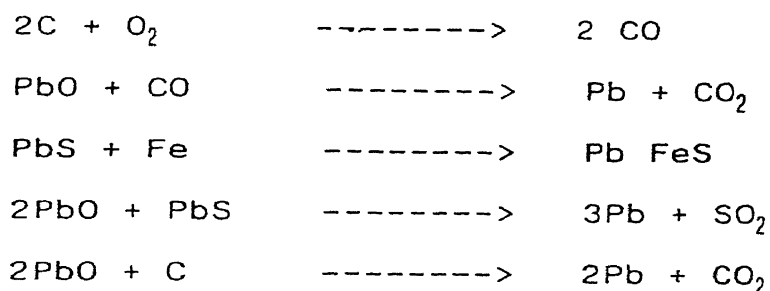
Blast furnace reaction

$2C + O_2 \longrightarrow 2\ CO$

$PbO + CO \longrightarrow Pb + CO_2$

$PbS + Fe \longrightarrow Pb\ FeS$

$2PbO + PbS \longrightarrow 3Pb + SO_2$

$2PbO + C \longrightarrow 2Pb + CO_2$

OBSERVED PARAMETERS IN BLAST FURNACE

Parameters	Time (Hours)			
	11.50	13.50	14.35	Average
Flue gas temperature °C	450	445	440	445.0
Air flow rate Nm^3/hr	6000	6000	5800	5933.0
Air pressure mm wg	1275	1300	1360	1316.6
Oxygen supply rate Nm^3/hr	50	50	50	50.0
Jacket water oulet temp.°C				
Mantel Jacket 1	49	45	47	47.0
Mantel Jacket 2	49	45	45	46.3
Mantel Jacket 3	49	45	47	47.0
Mantel Jacket 4	45	45	45	45.0
Mantel Jacket 5	49	45	47	47.0
Mantel Jacket 6	50	49	49	49.3
Mantel Jacket 7	52	50	50	50.6
Mantel Jacket 8	51	49	49	49.6
Mantel Jacket 9	50	49	49	49.3
Mantel Jacket 10	51	49	49	49.6
Mantel Jacket 11	51	47	49	49.0
Mantel Jacket 12	50	45	45	46.6
Main Jacket 1	52	51	53	51.0
Main Jacket 2	52	49	53	51.3

Appendix - 15.2/2 contd.

Parameters	Time (Hours)			
	11.50	13.50	14.35	Average
Main Jacket 3	45	54	53	50.6
Main Jacket 4	46	45	46	45.6
Main Jacket 5	57	57	57	57.0
Main Jacket 6	48	45	47	46.6
Main Jacket 7	61	56	57	58
Main Jacket 8	44	40	41	41.6
Main Jacket 9	59	55	55	56.3
Main Jacket 10	63	60	60	61
Main Jacket 11	52	59	48	53
Main Jacket 12	50	48	49	49
Main Jacket 13	44	45	44	44.3
Main Jacket 14	54	45	55	51.3
Main Jacket 15	53	53	52	52.66
Main Jacket 16	54	59	59	57.33
Main Jacket 17	52	50	51	51
Main Jacket 18	50	52	52	51.3
Cooling water outlet temp.in				
Chute	44	45	46	45
Channel	44	45	47	45.3
Lead temperature °C	1030	1100	1090	1073.3
Slag temperature °C	1350	1375	1350	1358.3
CO_2 in flue gas	17.5	17.5	17.5	17.5

No.of charges per shift = 16

Charge composition = 4 MT sinter
700 kg coke (hard)
150 kg of slag
40 kg of PboH

Total weight of charge = 4890 kgs

Total weight of charge per shift = 78240 kg/hr

Lead output/shift = 35% of total sinter input

= 0.35 x 4000 x 16

= 22400 kg/shift

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Appendix - 15.2/2 contd..

SUMMARY OF OBSERVED PARAMETERS

Parameters	Average
Flue gas temperature °C	445.0
Air flow rate Nm^3/hr	5933.0
Air pressure mm wg	1316.6
Oxygen flow rate Nm^3/hr	50.0
Mantel Jacket water outlet temp.°C	48.02
Main Jacket water outlet temp.°C	51.60
Chute Jacket water outlet temp.°C	45.0
Channel Jacket water outlet temp.°C	45.3
Lead temperature °C	1073.3
Slag temperature °C	1358.3
CO_2 in flue gas %	17.5

APPENDIX - 15.2/3

ENERGY BALANCE - BLAST FURNACE

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Total material input per hour

Sinter	= 8 MT = 8000 kgs
Coke	= 1400 kgs
Slag	= 300 kgs
PboH	= 80 kgs
Total input	= 9780 kg/hr

. Lead output = 35% total sinter input

= (0.35 x 8000)

= 2800 kg/hr

i. Slag output = 65% of total input feed

= 0.65 x 9780

= 6357 kg/hr

emental Composition of Slag

Element	Percentage	kg/hr
Lead	2	127.14
SiO_2	20	1271.40
Acid in solubles	23.5	1493.90
CaO	14	889.9
FeO	35	2224.9
Al_2O_3	5	317.85
Cu	0.5	31.78
Silver	0.003	0.19
Total		6357.00

. Heat Balance

Heat Input

ıt given through coke = 5500 x 1400
5500 kcal/kg

= 7700000 kcal/kg

Appendix - 15.2/3 contd..

b. Various heat outputs

1. Heat given to material
2. Heat given to endothermic reaction of PbO
3. Heat given to the slag
4. Heat given to cooling water (main, mantle, chute, channel and slag spent)
5. Heat loss due to flue gases
6. Surface heat losses
7. Heat loss due to CO in flue gases
8. Unaccounted losses.

1. Heat given to material

Lead output = 2800 kg/hr

Outlet lead temp. = 1073.3 °C

Lead melting point = 327°C

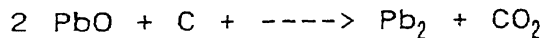
Latent heat of lead = 5.909 kcal/kg

Input lead temp. = 30°C

Heat given to lead = 2800 x [(0.03 (327-30) + 5.909+0.03 (1073.3-327)]

= 104182 kcal/hr

2. Heat given to endothermic reaction of PbO

$2\ PbO + C + \longrightarrow Pb_2 + CO_2$

2 (207+16) 12 207x2 44

$$1 \text{ kg of Pb required} \longrightarrow \frac{2\ (207+16)}{207 \times 2} \text{ kg of PbO}$$

= 1.077 kg of PbO

Considering the total lead component in the sinter supplied is in the form of PbO, this PbO reacted with in the furnace is given by :

PbO supply rate = 1.077 x output lead

= 1.077 x 2800 kg/hr

= 3015.60 kg/hr

Appendix - 15.2/3 contd..

Heat of reaction = 234.76 kcal/kg of PbO

Total heat supplied = 234.76 x 3015.6

= 707942 kcal/hr

3. Heat given to the slag material

Temperature of slag = 1358°C

Total slag output = 6357 kg/hr

Material	Quantity	Sp.heat	Latent heat	Heat absorbed kcal/h
Lead	127.14	0.03	5.909	5818
SiO_2	1271.40	0.32	-	540294
Acid in solubles	1493.90	0.23	-	456296
CaO	889.90	0.22	-	259993
Feo	2224.90	0.21	-	620480
Al_2O_3	317.85	0.27	-	113968
Cu	31.78	0.10	51.08	5843
Silver	0.19	0.05	25.04	17
	Total			2002709

4. Heat given to cooling water

Cooling water inlet temp. = 38.0°C

Jacket	Cooling water outlet temp °C	Flow rate kg/hr	Temp. raise °C	Heat given to cooling water kcal/hr
Mantle	48.02	27676	10.02	277313
Main	51.6	81234	13.61	1105594
Channel	45.3	2030	7.30	197319
Chute	45.0	15090	7.00	105630
Slag Spout	50.0	1500	12.0	18000
Total				1703856

Appendix - 15.2/3 contd.

5. Heat loss due to flue gases

Total flue gas quantity	= (air+oxygen+coke fuel) ash in coal
Air supplied	= 5933 Nm^3/hr
	= 7120 kg/hr
Oxygen supplied	= 50 m^3/hr
	= 50 x 1.42
	= 71 kg/hr
Fuel supplied	= 1400 kgs/hr
Ash in coke	= 12%
	= 1400 x 0.12
	= 168 kg/hr
Total flue gas quantity	= (7120 + 71 + 1400) -168
	= 8423 kg/hr
Flue gas temperature	= 445°C
Heat given to flue gases	= 0.24 (445-30) x 8423
	= 838930 kcal/hr

6. Surface heat losses

	Particulars	Area m^2	Temp °C	Rad Loss kcal/hr	Con Loss kcal/hr	Tot loss kcal/hr	kcal/hr per m^2
A.	**Bottom Base**						
	Front side	4 02	42	208 28	152 11	360 39	89 65
	Lead outlet side	6 40	50	574 76	458 60	1033.36	161 46
	Back Side	4 02	43	226 75	168 12	394.87	98 23
	Slag outlet side	6 40	52	638 45	516 62	1155.07	180.48
	Total (A)	20 84	-	1648 24	1295 45	2943 70	141 25
B.	**Tuyers Area**						
	Front side	2.90	41	137 06	98 42	235 48	81 20

Appendix - 15.2/3 contd..

	Particulars	Area m^2	Temp °C	Rad Loss kcal/hr	Con Loss kcal/hr	Tot loss kcal/hr	kcal/hr per m^2
	Lead outlet side	7.12	43	401.61	297 77	699 37	98 23
	Back Side	2 90	42	150 25	109 73	259 99	89 65
	Slag outlet side	7 12	44	434 63	326 67	761 29	106 92
	Total (B)	20 04	-	1123 54	832 59	1956 13	97 61
C.	First floor						
	Front side	6.40	135	4512 95	3644 44	8157 40	1274 59
	Lead outlet side	15 84	140	11971 02	9560 06	21531 08	1359 29
	Back Side	6 40	142	4969 64	3950 64	8920 28	1393 79
	Slag outlet side	15 84	150	13663 60	10658 51	24322 10	1535 49
	Total (C)	44 48	-	35117 21	27813 65	62930 86	1414 81
D.	Second floor						
	Front side	2 00	120	1128 27	939 29	2067 56	1033 78
	Lead outlet side	2 25	125	1371 12	1130 58	2501 71	1111 87
	Back Side	2 00	122	1164 06	965 45	2129 51	1064 76
	Slag outlet side	2 25	130	1476 85	1205 44	2682 29	1192 13
	Total (D)	8 50	-	5140 31	4240 76	9381 07	1103 66
	Grand Total (A+B+C+D)	93.86		43029.30	34182.46	77211 76	822.63

Total Surface heat losses = 77211 kcal/h

7. Heat loss due to unburnt CO in flue gases

Average % of CO in flue gases = 2% V/V

Weight of CO in flue gases :

Component	V/V	Mol.wt.	Percentage (x) M.wt	Percentage W/W
CO_2	17.5	44	770	24.96
N_2	80.0	28	2254	73.09
CO	2.0	30	60	1.95
Total			3084	100.00

Appendix - 15.2/3 contd..

Weight of CO in flue gas = Total flue gas x 0.0195

= 8423 x 0.0195

= 164.24 kg/hr

Heat loss due to unburnt CO in flue gases (2423 kcal/kg of CO) = 164.23 x 2423

= 397929 kcal/hr

ENERGY BALANCE

Particulars	kcal/hr	Percentage
Heat Input		
Heat given through fuel	7700000	100.00
Heat Output		
Heat given to lead	104182	1.35
Heat given to PbO to form lead	707942	9.19
Heat given to slag	2002709	26.01
Heat given to cooling water	1703856	22.12
Flue gas losses	838930	10.90
Surface heat losses	77211	1.00
Heat loss due to unburnts (CO_2) in exhaust gases	397929	5.17
Heat given for complex reactions and unaccounted losses	1867241	24.25

APPENDIX - 15.2/4

QUANTIFICATION OF COOLING WATER IN BLAST FURNACE

Cooling water is used in the jackets of main frame near tuyers, mantle, chute and channels.

Cooling Water Jackets

i. No of main jackets = 18
ii. No.of mantel jackets = 12
iii. No.of chutes = 2
iv. No of channels = 2
v. Slag spout = 1
vi. Measuring bucket capacity = 13.25 lts
(ϕ top 0.27m, bottom 0.23m height 0.27m)

i. Cooling water flow rate in Mantle Jackets

Jacket No.	Time taken to fill bucket sec.	Water flow rate lph
1	12.5	3816
2	20	2385
3	24	1988
4	27	1767
5	13	3669
6	40	1193
7	35	1363
8	21	2271

Average water flow rate per jacket = 2306 kg/hr

Total water flow rate in 12 mantle jackets = 27676 kg/hr

Appendix - 15.2/4 contd..

ii. Cooling water flow rate in Main Jackets

Jacket No.	Time taken to fill bucket sec.	Water flow rate lph
1	7.5	6360
2	7.0	6814
3	6.0	7950
4	14.5	3890
5	23.0	2074
6	14.0	3407
7	11.5	4148
8	8.0	5963
9	18.0	2650
10	15.0	3180
11	14.0	3407
12	11.0	4336

Average water flow rate per jacket = 4513 kg/hr

Total water flow rate in 18 main jackets = 81234 kg/hr

iii. Cooling water flow rate in Chute Jackets

Jacket No.	Time taken to fill bucket sec.	Water flow rate lph
1	6.0	7950
2	2.5	19080

Total water flow rate in 2 chute jackets = 27030 kg/hr

Appendix - 15.2/4 contd..

iv. Cooling water flow rate in Channel Jackets

Jacket No.	Time taken to fill bucket sec.	Water flow rate lph
1	6	7950
2	6	7950

Total water flow rate in 2 channel jackets = 15900 kg/hr

v. Cooling water flow rate in Slag Spout

Approximate water flow rate in slag spout = 1500 kg/hr

Total cooling water flow rate in Blast Furnace

Jacket	No.of Jackets	Water flow rate kg/hr
Mantle	12	27676
Main	18	81234
Chute	2	27030
Channel	2	15900
Slag	1	1500
Total		153340

Total cooling water flow rate = 153340 kg/hr

= 153.340 m^3/hr

APPENDIX - 15.2/5

COMBINED EFFICIENCY OF ROOTS BLOWER IN BLAST FURNACE

i. Blower function = supply of combustion through tuyer (ie., blast air)

ii. Blower air flow rate = 6000 m^3/hr

= 100 m^3/min

iii. Outlet air pressure = 1360 mmwg

iv. Actual power consn. = 52.62 kW

v. Theoretical power consn. = $\frac{m^3/min \times mmwg}{6120}$

$= \frac{100 \times 1360}{6120}$

= 22.22 kW

vi. Efficiency of the fan including blower = $\frac{22.22}{52.62}$

= 42%

APPENDIX - 15.3/1

MONTH-WISE ENERGY CONSUMPTION AND PRODUCTION IN SLAG SETTLER FOR THE YEAR 1994-95

Month	FO kL	Production MT	Operating hrs	Sp. FO consumption Lt/MT
Apr 94	33.00	1202.00	548:30	27.454
May	35.00	1060.00	533:35	33.019
Jun	15.00	508.00	393:45	29.528
Jul	24.00	1703.00	59:05	14.093
Aug	17.00	1002.00	415:05	16.966
Sep	21.00	1128.00	532:05	18.617
Oct	20.00	1054.00	563	18.975
Nov	25.00	926.00	476:35	26.998
Dec	32.00	1050.00	535:50	30.476
Jan 95	16.00	380.00	198:25	42.105
Feb	20.00	810.00	388:40	24.691
Mar	25.00	1220.00	497:25	20.492
Total	283.00	12043.00	-	25.285

Min. Specific energy consumption = 14.093 L/MT of lead

Max. Specific energy consumption = 42.105 L/MT of lead

Avg. Specific energy consumption = 25.285 L/MT of lead

APPENDIX - 15.3/2

OBSERVED PARAMETERS IN SLAG SETTLER

i.	No.of burners installed	= 2 Nos.
ii.	No.of burners in operation	= 2 Nos.
iii.	Fuel used	= FO
iv.	Oil pressure	= 7.4 kg/cm^2g
v.	Lead output	= 200 kg/shift
vi.	Slag input	= 6357 kg/hr
vii.	Frequency of slag input to the tank	= 3 times per hour
viii.	Flue gas temperature	= 940°C
ix.	Cooling water flow rate	= 9540 kg/hr
x.	Cooling water inlet temp.	= 37°C
xi.	Cooling water outlet temp.	= 52°C
xii.	Combustion air	
	Suction air velocity	= 11.5 m/sec
	Cross sectional area of suction	= 0.154 m^2
	Combustion air flow rate	= a x v
		= 11.5 x 0.154
		= 0.177 m^3/sec
		= 637 m^3/hr
		= 764 kg/hr
xiii.	Furnace oil consumption	= 51.5 lt/hr
		= 51.5 x 0.95
		= 48.95 kg/hr

Appendix - 15.3/2 contd..

Input slag composition to percentage slag settler (maximum values)

Lead	= 2.8%
Si	= 26.5%
FeO	= 36.0%
ZnO	= 6.8%
CaO	= 13.8%
Cu	= 0.15%
Ag	= 60 gpt
Al_2O_3	= 8%
Acid insolubles	= 30%

APPENDIX - 15.3/3

ENERGY BALANCE OF SETTLING TANK

i. Energy input :

Heat input through FO = 48.95 x 10200

= 499290 kcal/hr

ii. **Heat output :**

a. Flue gas losses
b. Cooling water losses
c. Heat loss due to openings
d. Surface heat losses
e. Useful heat to melt the slag + unaccounted losses

a. Flue gas losses

Flue gas quantity = Fuel + air

= 48.95 +764

= 812.5 kg/hr

Ambient temperature = 30°C

Flue gas losses = 0.25 x 812.5 (940-30)

= 184844 kcal/hr

b. Cooling water losses

= Rise in water temp x flow rate

= (52-37) x 9540

= 143100 kcal/h

Appendix - 15.3/3 contd..

c. Heat loss due to openings

Total radiation factor (TRF)

Opening	Height cm	Length cm	Width cm	Height/ width	Temp°C	TRF
Slag outlet	30	40	30	1.00	1200	0.6
Slag inlet	30	15	30	0.50	1200	0.5

Heat losses = TRF x Black body radiation x C/s area x Emissivity

Heat loss due to opening

Opening	Height cm	Length cm	Width cm	Ratio *	Temp°C	TRF
Slag outlet	40	40	30	1.3	1200	0.7
Slag inlet	30	15	30	0.5	1200	0.45

* Ratio = Height/width

** TRF = Total radiation factor

Heat losses = TRF x Black body radiation x C/s area x Emissivity

where, E = Emissivity (0.8)
BBR in kcal/h/m^2
area = area of opening

Heat losses

Opening	Area	BBR	Heat loss
Slag outlet	0.16	234000	20966
Slag inlet	0.045	234000	3790
Total			24756

Appendix - 15.3/3 contd..

Surface heat losses

	Particulars	Area m^2	Temp °C	Rad Loss kcal/h	Con Loss kcal/h	Tot loss kcal/h	kcal/h per m^2
A.	**Bottom Base**						
	Slag inlet side	2 90	170	3190.71	2366 04	5556.76	1916.12
	Front side	2.08	160	2032.80	1546 88	3579 68	1721.00
	Lead outlet side	2 90	158	2765 79	2115 32	4881 11	1683.14
	Slag outlet side	2.08	220	3852 42	2485.82	6338.24	3047.23
	Total (A)	9 96	-	11841 72	8514.06	20355 79	2043.75
B.	**Furnace area**						
	Slag inlet side	2 50	49	212 25	168 01	380 26	152 11
	Front side	1 80	48	144 07	113.06	257 13	142 85
	Lead outlet side	2 50	48	200.10	157.03	357 13	142 85
	Slag outlet side	1 80	48	144 07	113 06	257 13	142.85
	Top	2 47	90	807.04	908 08	1715 12	694 38
	Total (B)	11 07	-	1507 52	1459 26	2966 78	268 00
	Grand Total (A+B)	21.03	-	13349.25	9973.32	23322.57	1109.01

Total surface heat loss = 23327 kcal/h

HEAT BALANCE

Particulars	kcal/hr	Percentage
Heat Input	499290	100.00
Heat Output		
Flue gas losses	184844	37.02
Cooling water losses	143100	28.66
Losses due to opening	24756	4.96
Surface heat losses	23327	4.67
Efficiency & unaccounted losses	123263	24.69

APPENDIX - 15.4/1

BRIEF PROCESS DESCRIPTION OF LEAD REFINERY

No.of kettles in the process = 8

Kettle No	Brief Process description	Activity
1 & 2	The input lead ingots heated upto 400-450°C and saw dust is added The impurities will form as dross and the melt temp is reduced to 330°C (by natural cooling) The dry dross formed is removed For decopperisation sulphur is added to form wet dross which inturn converted to dry dross by adding saw dust and finally removed The outlet lead have copper below 300 GPT Residence time upto = 500 hours Fuel consumption (melting) = 6 7 lts/t Decopperisation time = 24 hours Fuel consumption during decopperisation = 5/t	Ordinary drossing & decopperisation
3	Standby - which was earlier used for desilverisation - I stage This desilverisation I stage is being carried out in kettle No 5	-
4	Molten metal from kettle No 1 or 2 is transferred to this kettle and heated upto 500°C Caustic soda flakes added and melt is agitated to form arsonate dross which is removed and metal is transferred to kettle No 5 Caustic soda addition = 0 5 kg/t Fuel required = 3 1 l/t Temperature range = 470-490°C Cycle time = 6 hours	Dearsonating
5	3-5 t of low silver (1-2% silver) crest pieces charged and agitated for 15-30 min The metal temp raised upto 450°C and the melt is cooled Once temp falls below 400°C crest forms This crest is removed till the temp reaches 340°C This crest is called high silver crest (silver 5%, Zinc 12%) The metal temp is raised upto 460°C (max) Metallic zinc (1200 kgs) added Once zinc is melted the temp of melt is brought down to 320°C The crest formed (having silver 20 GPT) is removed The molten metal is transferred to kettle No 6 Zinc addition = 11-12 5 kg/t Final silver content = 5-10 g/t Cycle time I stage = 7-9 hours II stage = 15-16 hours Fuel consumption I stage = 5 0 l/t II stage = 3 4 l/t Temperature = 460°C	Desilverisation - I stage Desilverisation -II stage

Appendix - 15.4/1 contd

Kettle No.	Brief Process description	Activity
6	The melt temp raised upto 595°C The zinc added in kettle No.5 is removed in this kettle by applying vacuum of 0 5mm for 6 hours During vacuum application agitation was done in anti-clockwise direction. Cycle time = 13 hours Zinc after dezincing = 0 05% Zinc recovery = 90% Fuel consumption = 6 4 l/t Temperature = 590°C Min vacuum = 0 5 um	Dezincing
7	The molten metal from kettle No.6 is transferred to this kettle for softening Caustic soda (NaOH) and Sodium Nitrate ($NaNO_3$) added During first stage of operation NaOH and $NaNO_3$ is added in the ratio of 200 100, while in the second stage the ratio is 150·125 In this kettle, the residual zinc and residual antimony is also removed Caustic soda addition = 8 4 kg/t Sodium Nitrate = 2 8 kg/t Fuel consumption = 9 0 l/t Temperature = 480-490°C	Softening
8	The molten metal from kettle 7 is transferred to this kettle The excess dross is removed and the metal temp is maintained above 450°C for moulding and to avoid solidifying during the transfer of metal from metal to casting machine The speed of casting machine is 10 t/hr Cycle time = 8 hours Temperature = 450°C (min) Weight of casting = 25 kg	Casting

APPENDIX - 15.4/2

MONTH-WISE ENERGY CONSUMPTION AND PRODUCTION IN LEAD REFINERY FOR THE YEAR 1994-95

Month	FO kL	LDO kL	Equi.FO for LDO, kL	Total FO, kL	Actual prodn, MT	Operating hrs	Sp.FO consn., l/MT
Apr 94	89.087	44.20	41.874	130.961	1260.298	468	103.912
May	70.413	7.20	6.821	77.234	532.654	328·45	144 999
Jun	57.435	69.90	66.221	123.656	989.525	473	124.965
Jul	62.272	94.26	89.299	151.571	1131.698	566:35	133.932
Aug	29.207	90.84	86.059	115.266	1090.305	596	105.719
Sep	27.576	128.52	121.756	149.332	945.325	457:10	157.969
Oct	71.958	96.04	90.985	162.943	760.037	435:45	214.389
Nov	37.046	78.48	74.349	111.395	749.639	373:35	148 599
Dec	44.000	51.74	49.017	93.017	828.144	388:20	112.320
Jan 95	9.068	58.00	54.947	64.015	784.670	297:15	81.583
Feb	12.214	54.74	51.859	64.073	154.710	196:45	414.149
Mar	10.000	61.08	57.865	67.865	915.773	320:30	74.107
Total	520.276	835.00	791.053	1311.329	10142.778	-	151.387

Min. Specific energy consumption = 414.149 l/MT of FO

Max. Specific energy consumption = 74.107 l/MT of FO

Avg. Specific energy consumption = 151.387 l/MT of FO

Conversion of LDO consumption into equivalent FO consumption $= \dfrac{\text{LDO (kL)} \times 0.85 \times 10800}{0.95 \times 10200}$

APPENDIX - 15.4/3

REFINING KETTLE COMBUSTION EFFICIENCY CALCULATIONS

DATA : (Basis - per kg of fuel)

Procedure	= BIS8753
Ambient air temperature	= Ta
Wet bulb temperature	= Tw
Flue gas temperature	= Tf
Specific humidity	= SH kg/kg of air
FO specific heat	= 0.5 kcal/kg°C
Furnace oil temperature	= Tfa
CO_2 in flue gas	= Act CO_2
Furnace oil calorific value	= 10200 kcal/kg
Furnace oil consumption	= F kg/h

Furnace oil ultimate analysis (weight basis)

Carbon (C)	= 84.00%
Hydrogen (H)	= 11.50%
Sulphur (S)	= 3.50%
Moisture (H_2O)	= 1.00%

Analysis

A. Oxygen requirement for combustion = 3.198 kg

Combustion air requirement = 13.78 kg

Flue gas quantity (Stoichiometric) :

a. **Weight Basis**

CO_2	= 3.083 kg
H_2O from H_2	= 1.038 kg
H_2O from moisture in fuel	= 0.010 kg

Appendix - 15.4/3 contd..

SO_2	= 0.070 kg
N_2 in combustion air	= 10.586 kg
Total flue gas	= 14.784 kg
Dry flue gas quantity	= 13.739 kg
Maximum CO_2 in flue gas	= 22.439 wt/wt

b. Volume basis (at STP)

CO_2	= 1.562 m^3
H_2O from H_2	= 1.283 m^3
H_2O from moisture in fuel	= 0.012 m^3
SO_2	= 0.024 m^3
N_2 in combustion air	= 8.424 m^3
Total flue gas	= 11.306 m^3
Dry flue gas quantity	= 10.010 m^3
Maximum CO_2 in flue gas	= 15.608 v/v

B. % Excess air in flue gas (EA) $= \left(\frac{\text{Max } CO_2}{\text{Act } CO_2} - 1 \right) \times 100$

C. Total air supplied (TA) $= 13.784 \times \left(\frac{EA}{100} + 1 \right)$ kg

D. Total flue gas = (TA+1) kg

E. Actual H_2O vapour due to vapour in combustion air = SH x TA kg

F. Heat Balance

* Heat Input :

a. Heat value of the fuel = 10200 kcal ---------> (a)

b. Sensible heat in fuel = 0.5 (Tfa-Ta) kcal --->(b)

Total heat input (THI) = a+b kcal

Appendix - 15.4/3 contd..

* **Heat Output**

c. Flue gas losses (kcal)

$$= \frac{100}{12 \times CO_2} \left(\frac{C}{100} + \frac{S}{207} \right) \times 30.6\ (Tf-Ta) \times \frac{1}{4.18} \quad \text{--->(c)}$$

d. Heat loss due to H_2 in fuel (kcal)

$$= \frac{9xH}{100} \times (1.88\ (Tf-Ta) + 2442) \times \frac{1}{4.18} \quad \text{------> (d)}$$

e. Heat loss due to H_2O in fuel (kcal)

$$= \frac{H_2O}{100} (1.88\ (Tf-Ta) + 2442) \times \frac{1}{4.18} \quad \text{------> (e)}$$

f. Heat loss due to moisture in air

$$kcal = [\ TA \times SH \times 1.88\ (Tf-Ta)\ ] \times \frac{1}{4.18} \quad \text{-----> (f)}$$

g. Surface heat losses

* Radiation losses (kcal/hr)

$= 5.67 \times 10^{-8} \times 0.7\ [(Ts+273)^4 - (Ta+273)^4] \times 0.86 \times$ surface area $(m^2$

* Convection losses (kcal/hr)

$= C \times (Ts-Ta)^{1.25} \times 0.86 \times$ surface area

C = 2.56 upward facing horizontal hot surface

C = 1.97 flat vertical surfaces

Surface heat loss = Radiation loss + Convection loss (kcal/hr)

$$\text{Surface heat loss Per kg of fuel} = \frac{\text{(radiation loss+convection loss)}}{\qquad} \quad \text{------> (g}$$

Appendix - 15.4/3 contd..

h. Heat loss due to furnace door opening

(Refer Appendix - 15.4/6 for details)

HEAT BALANCE

Particulars	kcal	%
Heat input		
a. Heat given through fuel	(a)	(a/THI)x100
b. Sensible heat in fuel	(b)	(b/THI)x100
Total heat input (THI)	(a+b)	100.00
Heat Output		
c. Flue gas losses	(c)	(c/THI)x100
d. Heat loss due to H_2 in fuel	(d)	(d/THI)x100
e. Heat loss due to H_2O in fuel	(e)	(e/THI)x100
f. Heat loss due to H_2O in air	(f)	(f/THI)x100
g. Surface heat losses	(g)	(g/THI)x100
h. Heat loss due to door opening	(h)	(h/THI)x100
i. Useful heat + unaccounted losses (a + b) - (c + d + e + f + g + h)	(i)	(i/THI)x100

Appendix - 15.4/3 contd..

COMBUSTION EFFICIENCY CALCULATIONS - REFINING KETTLE

Kettle No.1

Basis Per kg of fuel
Procedure BIS 8753

DATA

Type of fuel	=	Furnace Oil
Fuel consumption rate	=	123.5 kg/hr
Flue gas temperature	=	402 °C
CO_2 in flue gas	=	3 %
Material temperauture	=	365°C
Ambient temperature	=	30 °C
Wet bulb temperature	=	28 °C
Moisture content	=	0.023 kg/kg of air
Fuel calorific value	=	10200 kcal/kg
Fuel input temperature	=	75 °C
Specific heat of fuel	=	0.5 kcal/kg °C
Temperature of combustion air	=	30 °C

ANALYSIS

Excess air in flue gas %	=	420.28
Combustion air requirement	=	13.78 kg
Total air supplied	=	71.71 kg
Total flue gas quantity	=	72.71 kg
Excess air quantity	=	57.93 kg
Heat loss due to excess air	=	4525.46 kcal
H_2O vapour in flue gas		
Due to H_2 in fuel	=	1.04 kg
Due to H_2O in fuel	=	0.01 kg
Due to H_2O in air	=	1.65 kg
Dry flue gas quantity	=	70.02 kg

Appendix - 15.4/3 contd..

Surface Losses

Particulars	Area m2	Temp Deg C	Rad Loss kCal/hr	Con Loss kCal/hr	Tot loss kCal/hr	kCal/hr per m2
A. Bottom Cylindrical Portion						
Door side	2 56	75	467 16	505 50	972.66	379 95
Left side to the door	2 56	78	505 53	547 97	1053.50	411 52
Back Side to the door	2 56	83	571 71	620 23	1191 93	465.60
Right side to the door	2 56	95	742 30	800 47	1542 77	602 65
Total (A)	10 24	-	2286 71	2474 17	4760 87	464 93
B. Cylindrical portion -I						
Door side	4 50	87	1101 61	1194 05	2295 66	510 15
Left side to the door	4 50	100	1438 75	1543 65	2982 40	662 76
Back Side to the door	4 50	90	1176 25	1273 11	2449 36	544 30
Right side to the door	4 50	95	1304 83	1407 08	2711 91	602 65
Total (B)	18 00	-	5021 44	5417 89	10439 33	579 96
C. Cylindrical portion -II						
Door side	4 50	79	911 50	988 38	1899 88	422 20
Left side to the door	4 50	76	843 48	913 32	1756 80	390 40
Back Side to the door	4.50	80	934 57	1013 65	1948 23	432 94
Right side to the door	4 50	74	799 09	863 96	1663 04	369 57
Total (C)	18 00	-	3488 64	3779 31	7267 95	403 77
D Burner block (E=.75)	1 50	125	914 08	753 72	1667 80	1111 87
E Top horizontal surface (E= 75)	7 06	300	25657 08	17011 70	42668 79	6043 74
Grand Total (A+B+C+D+E)	28 24	-	35081 24	26962 63	62043 87	2197 02

Emissivity = .6

Total surface heat losses = 62043 kcal/hr

Surface losses per kg of FO supplied = 502.37 kcal

Appendix - 15.4/3 contd..

Heat loss due to door opening
(Refer Appendix - 15.4/6 for details)

Total black body radiation = 14152 kcal/hr

Radiation loss per kg of = 114.59 kcal

HEAT BALANCE

Particulars	kcal/kg	Percentage
Heat input		
Through heat value of oil	10200.00	99.78
Sensible heat in fuel	22.50	0.22
Total heat input	10222.50	100.00
Heat output		
Flue gas losses	6453.42	63.13
Heat loss due to H_2 in fuel	777.82	7.61
Heat loss due to moisture in fuel	7.52	0.07
Heat loss due moisture in air	294.52	2.88
Surface heat loss	502.37	4.91
Heat loss through door opening	114.59	1.12
Useful heat + unaccounted losses	2072.26	20.27
Total	10223.00	100.00

Appendix - 15.4/3 contd..

Kettle No.2

Basis Per kg of fuel
Procedure BIS 8753

DATA

Type of fuel	=	Furnace Oil
Fuel consumption rate	=	127.5 kg/hr
Flue gas temperature	=	391 °C
CO_2 in flue gas	=	3%
Material temperature	=	460 °C
Ambient temperature	=	30 °C
Wet bulb temperature	=	28 °C
Moisture content	=	0.023 kg/kg of air
Fuel calorific value	=	10200 kcal/kg
Fuel input temperature	=	75 °C
Specific heat of fuel	=	0.5 kcal/kg °C
Temperature of combustion air	=	30 °C

ANALYSIS

Excess air in flue gas %	=	420.28
Combustion air requirement	=	13.78 kg
Total air supplied	=	71.71 kg
Total flue gas quantity	=	72.71 kg
Excess air quantity	=	57.93 kg
Heat loss due to excess air	=	4391.64 kcal
H_2O vapour in flue gas		
Due to H_2 in fuel	=	1.04 kg
Due to H_2O in fuel	=	0.01 kg
Due to H_2O in air	=	1.65 kg
Dry flue gas quantity	=	70.02 kg

Appendix - 15.4/3 contd..

Surface Losses

Particulars	Area m2	Temp Deg C	Rad Loss kCal/hr	Con Loss kCal/hr	Tot loss kCal/hr	kCal/hr per m2
A. Bottom Cylindrical Portion						
Door side	2 56	80	531 67	576.66	1108 32	432 94
Left side to the door	2 56	100	818 49	878 17	1696.66	662 76
Back Side to the door	2 56	82	558 25	605 63	1163 88	454.64
Right side to the door	2 56	81	544 90	591 11	1136 01	443 75
Total (A)	10.24	-	2453 31	2651 56	5104 87	498 52
B. Cylindrical portion -I						
Door side	4 50	120	2030 89	2113 39	4144 29	920 95
Left side to the door	4 50	100	1438 75	1543.65	2982 40	662 76
Back Side to the door	4 50	80	934 57	1013 65	1948 23	432 94
Right side to the door	4 50	70	712 58	766 92	1479.51	328 78
Total (B)	18 00	-	5116 80	5437.62	10554 42	586 36
C. Cylindrical portion -II						
Door side	4 50	100	1438 75	1543 65	2982 40	662 76
Left side to the door	4.50	110	1723 23	1824 06	3547.30	788 29
Back Side to the door	4 50	102	1493 85	1598 98	3092 83	687 30
Right side to the door	4 50	95	1304 83	1407 08	2711.91	602.65
Total (C)	18 00	-	5960 66	6373 78	12334 44	685 25
D. Burner block (E= 75)	1 50	150	1293 90	1009.33	2303 23	1535.4
E. Top horizontal surface (E=.75)	7 06	360	39277 31	21861 78	61139 09	8659 9
Grand Total (A+B+C+D+E)	28.24	-	51648.67	34682.51	86331.18	3057.0

Emissivity = 6

Total surface heat losses = 86331 kcal/hr

Surface losses per kg of FO supplied = 678.17 kcal

Heat loss due to door opening
(Refer Appendix - 15.4/6 for details)

Total black body radiation = 18869 kcal/hr

Radiation loss per kg of = 148.22 kcal

Appendix - 15.4/3 contd..

HEAT BALANCE

Particulars	kcal/kg	Percentage
Heat input		
Through heat value of oil	10200.00	99.78
Sensible heat in fuel	22.50	0.22
Total heat input	10222.50	100.00
Heat output		
Flue gas losses	6262.59	61.26
Heat loss due to H_2 in fuel	772.70	7.56
Heat loss due to moisture in fuel	7.47	0.07
Heat loss due moisture in air	285.81	2.80
Surface heat loss	678.17	6.63
Heat loss through door opening	148.22	1 45
Useful heat + unaccounted losses	2067.53	20.23
Total	10223.00	100.00

Appendix - 15.4/3 contd..

Kettle No.4

Basis Per kg of fuel
Procedure BIS 8753

DATA

Type of fuel	=	Furnace Oil
Fuel consumption rate	=	123.5 kg/hr
Flue gas temperature	=	590 °C
CO_2 in flue gas	=	13%
Material temperature	=	750 °C
Ambient temperature	=	30 °C
Wet bulb temperature	=	28 °C
Moisture content	=	0.023 kg/kg of air
Fuel calorific value	=	10200 kcal/kg
Fuel input temperature	=	75 °C
Specific heat of fuel	=	0.5 kcal/kg °C
Temperature of combustion air	=	30 °C

ANALYSIS

Excess air in flue gas %	=	20.06
Combustion air requirement	=	13.78 kg
Total air supplied	=	16.55 kg
Total flue gas quantity	=	17.55 kg
Excess air quantity	=	2.77 kg
Heat loss due to excess air	=	325.23 kcal
H_2O vapour in flue gas		
Due to H_2 in fuel	=	1.04 kg
Due to H_2O in fuel	=	0.01 kg
Due to H_2O in air	=	1.38 kg
Dry flue gas quantity	=	16.12 kg

Appendix - 15.4/3 contd..

Surface Losses

Particulars	Area m2	Temp Deg C	Rad Loss kCal/hr	Con Loss kCal/hr	Tot loss kCal/hr	kCal/hr per m2
A. Bottom Cylindrical Portion						
Door side	2 56	100	818 49	878 17	1696 66	662 76
Left side to the door	2 56	62	311 99	330 10	642 09	250 82
Back Side to the door	2 56	55	235 59	242 45	478 04	186 73
Right side to the door	2 56	90	669 16	724 26	1393 41	544 30
Total (A)	10 24	-	2035 23	2174 98	4210 20	411 15
B. Cylindrical Portion -I						
Door side	4 50	120	2030 89	2113 39	4144 29	920 95
Left side to the door	4 50	100	1438 75	1543 65	2982 40	662 76
Back Side to the door	4 50	80	934 57	1013 65	1948 23	432 94
Right side to the door	4 50	90	1176 25	1273 11	2449 36	544 30
Total (B)	18 00	-	5580 47	5943 81	11524 28	640 24
C. Cylindrical portion -II						
Door side	4 50	100	1438 75	1543 65	2982 40	662 76
Left side to the door	4 50	110	1723 23	1824 06	3547 30	788 29
Back Side to the door	4 50	102	1493 85	1598 98	3092 83	687 30
Right side to the door	4 50	95	1304 83	1407 08	2711 91	602 65
Total (C)	18 00	-	5960 66	6373 78	12334 44	685 25
D. Burner block (E=.75)	1 50	150	1293 90	1009 33	2303 23	1535 49
E. Top horizontal surface (E=.75)	7 06	550	116276 91	38596 44	154873 35	21936 74
Grand Total (A+B+C+D+E)	28 24	-	129111 94	51923.36	181035 30	6410 60

Emissivity = .6

Total surface heat losses = 181035 kcal/hr

Surface losses per kg of FO supplied = 1456.87 kcal

Heat loss due to door opening
(Refer Appendix - 15.4/6 for details)

Total black body radiation = 18869 kcal/hr

Radiation loss per kg of = 152.79 kcal

Appendix - 15.4/3 contd..

HEAT BALANCE

Particulars	kcal/kg	Percentage
Heat input		
Through heat value of oil	10200.00	99.78
Sensible heat in fuel	23.00	0.22
Total heat input	10223.00	100.00
Heat output		
Flue gas losses	2241.88	21.93
Heat loss due to H_2 in fuel	865.34	8.47
Heat loss due to moisture in fuel	8.36	0.08
Heat loss due moisture in air	102.31	1.00
Surface heat loss	1465.87	14.34
Heat loss through door opening	152.79	1.49
Useful heat + unaccounted losses	5385.95	52.69
Total	10223.00	100.00

Appendix - 15.4/3 contd..

Kettle No.6

Basis Per kg of fuel
Procedure BIS 8753

DATA

Type of fuel	=	Furnace Oil
Fuel consumption rate	=	123.5 kg/hr
Flue gas temperature	=	456 °C
CO_2 in flue gas	=	4%
Material temperature	=	365 °C
Ambient temperature	=	30 °C
Wet bulb temperature	=	28 °C
Moisture content	=	0.023 kg/kg of air
Fuel calorific value	=	10200 kcal/kg
Fuel input temperature	=	75 °C
Specific heat of fuel	=	0.5 kcal/kg °C
Temperature of combustion air	=	30 °C

ANALYSIS

Excess air in flue gas %	=	290.21
Combustion air requirement	=	13.78 kg
Total air supplied	=	53.78 kg
Total flue gas quantity	=	54.78 kg
Excess air quantity	=	40.00 kg
Heat loss due to excess air	=	3578.52 kcal
H_2O vapour in flue gas		
Due to H_2 in fuel	=	1.04 kg
Due to H_2O in fuel	=	0.01 kg
Due to H_2O in air	=	1.24 kg
Dry flue gas quantity	=	52.50 kg

Appendix - 15.4/3 contd..

Surface Losses

Particulars	Area m2	Temp Deg C	Rad Loss kCal/hr	Con Loss kCal/hr	Tot loss kCal/hr	kCal/hr per m2
A. Bottom Cylindrical Portion						
Door side	2 56	65	346 24	369.22	715 46	279 48
Left side to the door	2 56	68	381 41	409 20	790 61	308 83
Back Side to the door	2 56	70	405 38	436 29	841 67	328 78
Right side to the door	2 56	68	381 41	409.20	790 61	308 83
Total (A)	10 24	-	1514 44	1623 92	3138 36	306 48
B Cylindrical portion -I						
Door side	4 50	63	568 31	603 00	1171 32	260 29
Left side to the door	4 50	74	799 09	863 96	1663 04	369 57
Back Side to the door	4 50	75	821 18	888 57	1709 76	379 95
Right side to the door	4 50	75	821 18	888 57	1709 76	379 9
Total (B)	18 00	-	3009 77	3244 11	6253 88	347 4
C. Cylindrical portion -II						
Door side	4 50	59	489 82	513 07	1002 89	222 8
Left side to the door	4 50	90	1176 25	1273 11	2449 36	544.3
Back Side to the door	4 50	58	470 64	491 05	961 69	213 7
Right side to the door	4 50	56	432 79	447 60	880.39	195 6
Total (C)	18 00	-	2569 50	2724 83	5294 33	294 1
D. Burner block (E=.75)	1 50	140	1133 62	905 31	2038 93	1359.
E Top horizontal surface (E=.75)	7 06	415	55673 42	26507 52	82180 94	11640
Grand Total (A+B+C+D+E)	28.24	-	62386.31	33381.76	95768.07	3391.

Emissivity = 6

Total surface heat losses	= 95768 kcal/hr
Surface losses per kg of FO supplied	= 775.45 kcal
Heat loss due to door opening (Refer Appendix - 15.4/6 for details)	
Total black body radiation	= 15095 kcal/hr
Radiation loss per kg of	= 122.23 kcal

Appendix - 15.4/3 contd..

HEAT BALANCE

Particulars	kcal/kg	Percentage
Heat input		
Through heat value of oil	10200.00	99.78
Sensible heat in fuel	23.00	0.22
Total heat input	10223.00	100.00
Heat output		
Flue gas losses	5542.63	54.22
Heat loss due to H_2 in fuel	802.96	7 85
Heat loss due to moisture in fuel	7.76	0.08
Heat loss due moisture in air	252.93	2.47
Surface heat loss	775 45	7.59
Heat loss through door opening	122.23	1.20
Useful heat + unaccounted losses	2718.50	26.59
Total	10223.00	100.00

Appendix - 15.4/3 contd..

Kettle No.7

Basis Per kg of fuel
Procedure BIS 8753

DATA

Type of fuel	=	Furnace Oil
Fuel consumption rate	=	123.5 kg/hr
Flue gas temperature	=	522 °C
CO_2 in flue gas	=	6%
Material temperature	=	630 °C
Ambient temperature	=	30 °C
Wet bulb temperature	=	28 °C
Moisture content	=	0.023 kg/kg of air
Fuel calorific value	=	10200 kcal/kg
Fuel input temperature	=	75 °C
Specific heat of fuel	=	0.5 kcal/kg °C
Temperature of combustion air	=	30 °C

ANALYSIS

Excess air in flue gas %	=	160.14
Combustion air requirement	=	13.78 kg
Total air supplied	=	35.86 kg
Total flue gas quantity	=	36.86 kg
Excess air quantity	=	22.07 kg
Heat loss due to excess air	=	2280.58 kcal
H_2O vapour in flue gas		
Due to H_2 in fuel	=	1.04 kg
Due to H_2O in fuel	=	0.01 kg
Due to H_2O in air	=	0.82 kg
Dry flue gas quantity	=	34.99 kg

Appendix - 15.4/3 contd..

Surface Losses

Particulars	Area m2	Temp Deg C	Rad Loss kCal/hr	Con Loss kCal/hr	Tot loss kCal/hr	kCal/hr per m2
A. Bottom Cylindrical Portion						
Door side	2.56	60	289 67	304 51	594 18	232 10
Left side to the door	2.56	95	742 30	800 47	1542 77	602 65
Back Side to the door	2 56	72	429 77	463 73	893 50	349 02
Right side to the door	2 56	67	369 58	395 78	765 37	298 97
Total (A)	10 24	-	1831 32	1964 50	3795 82	370 69
B Cylindrical portion -I						
Door side	4 50	111	1752 94	1852 61	3605 54	801 23
Left side to the door	4 50	90	1176 25	1273 11	2449 36	544 30
Back Side to the door	4 50	96	1331 18	1434 19	2765 37	614 53
Right side to the door	4 50	99	1411 53	1516 14	2927 67	650 59
Total (B)	18 00	-	5671 90	6076 05	11747 94	652 66
C Cylindrical portion -II						
Door side	4 50	115	1874 09	1967 66	3841 75	853 72
Left side to the door	4 50	108	1664 52	1767 24	3431 76	762 61
Back Side to the door	4 50	118	1967 45	2054 85	4022 30	893 84
Right side to the door	4 50	107	1635 51	1738 96	3374 47	749 88
Total (C)	18 00	-	7141 57	7528 72	14670 29	815 02
D. Burner block (E=.75)	1 50	165	1556 59	1169 43	2726 01	1817 34
E Top horizontal surface (E= 75)	7 06	425	59110 82	27370 93	86481 74	12249 54
Grand Total (A+B+C+D+E)	28 24	-	73480 87	42145 12	115625 99	4094 40

Emissivity = 6

Total surface heat losses	= 115625 kcal/hr
Surface losses per kg of FO supplied	= 936.23 kcal
Heat loss due to door opening (Refer Appendix - 15.4/6 for details)	
Total black body radiation	= 15095 kcal/hr
Radiation loss per kg of	= 122.23 kcal

Appendix - 15.4/3 contd..

HEAT BALANCE

Particulars	kcal/kg	Percentage
Heat input		
Through heat value of oil	10200.00	99.78
Sensible heat in fuel	23.00	0.22
Total heat input	10223.00	100.00
Heat output		
Flue gas losses	4267.58	41.75
Heat loss due to H_2 in fuel	833.69	8.16
Heat loss due to moisture in fuel	8.05	0.08
Heat loss due moisture in air	194.76	1.91
Surface heat loss	936.23	9.16
Heat loss through door opening	122.23	1.20
Useful heat + unaccounted losses	3859.95	37.76
Total	10223.00	100.00

Appendix - 15.4/3 contd..

Kettle No.8

Basis	Per kg of fuel
Procedure	BIS 8753

DATA

Type of fuel	=	Furnace Oil
Fuel consumption rate	=	128.5 kg/hr
Flue gas temperature	=	520°C
CO_2 in flue gas	=	4%
Material temperature	=	500 °C
Ambient temperature	=	30 °C
Wet bulb temperature	=	28 °C
Moisture content	=	0.023 kg/kg of air
Fuel calorific value	=	10200 kcal/kg
Fuel input temperature	=	75 °C
Specific heat of fuel	=	0.5 kcal/kg °C
Temperature of combustion air	=	30 °C

ANALYSIS

Excess air in flue gas %	=	290.21
Combustion air requirement	=	13.78 kg
Total air supplied	=	33 78 kg
Total flue gas quantity	=	54.78 kg
Excess air quantity	=	40.00 kg
Heat loss due to excess air	=	4116.13 kcal
H_2O vapour in flue gas		
Due to H_2 in fuel	=	1.04 kg
Due to H_2O in fuel	=	0.01 kg
Due to H_2O in air	=	1.24 kg
Dry flue gas quantity	=	52.50 kg

Appendix - 15.4/3 contd..

Surface Losses

Particulars	Area m^2	Temp °C	Rad Loss kcal/hr	Con Loss kcal/hr	Tot loss kcal/hr	kCal/hr per m^2
A. Bottom Cylindrical Portion						
Door side	2 56	58	267 74	279 35	547 09	213.71
Left side to the door	2 56	66	357 86	382.46	740.32	289.19
Back Side to the door	2 56	69	393 34	422.70	816 05	318.77
Right side to the door	2 56	69	393 34	422 70	816 05	318.77
Total (A)	10 24	-	1412 29	1507 22	2919 50	285.11
B. Cylindrical portion -I						
Door side	4 50	77	865 96	938 21	1804 17	400 9[illegible]
Left side to the door	4 50	84	1028.82	1116 01	2144 83	476 6[illegible]
Back Side to the door	4 50	76	843 48	913 32	1756 80	390.4[illegible]
Right side to the door	4 50	74	799 09	863 96	1663 04	369.5
Total (B)	18 00	-	3537 34	3831 50	7368 84	409.3[illegible]
C. Cylindrical portion -II						
Door side	4 50	60	509 18	535 28	1044 46	232 1
Left side to the door	4 50	65	608 63	649 03	1257.65	279.4
Back Side to the door	4 50	62	548 43	580 25	1128 68	250.8
Right side to the door	4 50	64	588.38	625 93	1214 31	269.8
Total (C)	˙8 00	-	2254 61	2390 48	4645.10	258 0
D. Burner block (E=.75)	1 50	120	846 21	704 46	1550.67	1033 7
E Top horizontal surface (E=.75)	7 06	400	50790 99	25222 91	76013 91	10766.8
Grand Total (A+B+C+D+E)	28 24	-	57429 15	32149 37	89578.51	3172.0

Emissivity = 6

Total surface heat losses	= 89578 kcal/hr
Surface losses per kg of FO supplied	= 697.11 kcal
Heat loss due to door opening (Refer Appendix - 15.4/6 for details)	
Total black body radiation	= 15282 kcal/hr
Radiation loss per kg of	= 118.93 kcal

Appendix - 15.4/3 contd..

HEAT BALANCE

Particulars	kcal/kg	Percentage
Heat input		
Through heat value of oil	10200.00	99.78
Sensible heat in fuel	23.00	0.22
Total heat input	10223.00	100.00
Heat output		
Flue gas losses	6375.35	62.37
Heat loss due to H_2 in fuel	832.75	8.15
Heat loss due to moisture in fuel	8.05	0.08
Heat loss due moisture in air	290.95	2.85
Surface heat loss	697.11	6.82
Heat loss through door opening	118.93	1.16
Useful heat + unaccounted losses	1899.36	18.58
Total	10223.00	100.00

APPENDIX - 15.4/4

COMBUSTION EFFICIENCY EVALUATION OF KETTLES AFTER CONTROLLING EXCESS AIR

Kettle No.1

Basis Per kg of fuel
Procedure BIS 8753

DATA

Fuel consumption rate	=	123.5 kg/hr
Flue gas temperature	=	402 °C
CO_2 in flue gas	=	12.5 %
Material temperauture	=	365°C
Ambient temperature	=	30 °C
Wet bulb temperature	=	28 °C
Moisture content	=	0.023 kg/kg of air
Fuel calorific value	=	10200 kcal/kg
Fuel input temperature	=	75 °C
Specific heat of fuel	=	0.5 kcal/kg °C
Temperature of combustion air	=	30 °C

ANALYSIS

Excess air in flue gas %	=	24.87
Combustion air requirement	=	13.78 kg
Total air supplied	=	71.21 kg
Total flue gas quantity	=	18.21 kg
Excess air quantity	=	3.43 kg
Heat loss due to excess air	=	267.76 kcal
H_2O vapour in flue gas		
Due to H_2 in fuel	=	1.04 kg
Due to H_2O in fuel	=	0.01 kg
Due to H_2O in air	=	0.40 kg
Dry flue gas quantity	=	16.77 kg

Appendix - 15.4/4 contd..

Surface Heat Losses

Total surface heat losses = 62043 kcal/hr

Surface losses per kg of FO supplied = 502.37 kcal

Heat loss due to door opening
(Refer Appendix - 15.4/6 for details)

Total black body radiation = 14152 kcal/hr

Radiation loss per kg of = 114.59 kcal

HEAT BALANCE

Particulars	kcal/kg	Percentage
Heat input		
Through heat value of oil	10200.00	99.78
Sensible heat in fuel	22.50	0.22
Total heat input	10222.50	100.00
Heat output		
Flue gas losses	1548.82	15.15
Heat loss due to H_2 in fuel	777.82	7.61
Heat loss due to moisture in fuel	7.52	0.07
Heat loss due moisture in air	70.68	0.69
Surface heat loss	502.37	4.91
Heat loss through door opening	114.59	1.12
Useful heat + unaccounted losses	7200.69	70.44
Total	10222.50	100.00

Appendix - 15.4/4 contd..

Kettle No.2

Basis Per kg of fuel
Procedure BIS 8753

DATA

Fuel consumption rate	=	127.5 kg/hr
Flue gas temperature	=	391 °C
CO_2 in flue gas	=	12.5%
Material temperature	=	460 °C
Ambient temperature	=	30 °C
Wet bulb temperature	=	28 °C
Moisture content	=	0.023 kg/kg of air
Fuel calorific value	=	10200 kcal/kg
Fuel input temperature	=	75 °C
Specific heat of fuel	=	0.5 kcal/kg °C
Temperature of combustion air	=	30 °C

ANALYSIS

Excess air in flue gas %	=	24.87
Combustion air requirement	=	13.78 kg
Total air supplied	=	17.21 kg
Total flue gas quantity	=	18.21 kg
Excess air quantity	=	3.48 kg
Heat loss due to excess air	=	259.84 kcal
H_2O vapour in flue gas		
Due to H_2 in fuel	=	1.04 kg
Due to H_2O in fuel	=	0.01 kg
Due to H_2O in air	=	0.40 kg
Dry flue gas quantity	=	16.77 kg

Appendix - 15.4/4 contd..

Surface Losses

Total surface heat losses = 86331 kcal/hr

Surface losses per kg of FO supplied = 678.17 kcal

Heat loss due to door opening
(Refer Appendix - 15.4/6 for details)

Total black body radiation = 18869 kcal/hr

Radiation loss per kg of = 148.22 kcal

HEAT BALANCE

Particulars	kcal/kg	Percentage
Heat input		
Through heat value of oil	10200.00	99.78
Sensible heat in fuel	22.50	0.22
Total heat input	10222.50	100.00
Heat output		
Flue gas losses	1503.02	14.70
Heat loss due to H_2 in fuel	772.70	7.56
Heat loss due to moisture in fuel	7.47	0.07
Heat loss due moisture in air	68.59	0.67
Surface heat loss	678.17	6.63
Heat loss through door opening	148.22	1.45
Useful heat + unaccounted losses	7044.32	68.91
Total	10222.50	100.00

Appendix - 15.4/4 contd..

Surface Losses

Total surface heat losses = 95768 kcal/hr

Surface losses per kg of FO supplied = 775.45 kcal

Heat loss due to door opening
(Refer Appendix - 15.4/6 for details)

Total black body radiation = 15095 kcal/hr

Radiation loss per kg of = 122.23 kcal

HEAT BALANCE

Particulars	kcal/kg	Percentage
Heat input		
Through heat value of oil	10200.00	99.78
Sensible heat in fuel	23.00	0.22
Total heat input	10223.00	100.00
Heat output		
Flue gas losses	1773.65	17.35
Heat loss due to H_2 in fuel	802.96	7.85
Heat loss due to moisture in fuel	7.76	0.08
Heat loss due moisture in air	80.94	0.79
Surface heat loss	775.45	7.59
Heat loss through door opening	122.23	1.20
Useful heat + unaccounted losses	6659.51	65.15
Total	10223.00	100.00

Kettle No.7

Basis Per kg of fuel
Procedure BIS 8753

DATA

Fuel consumption rate	=	123.5 kg/hr
Flue gas temperature	=	522 °C
CO_2 in flue gas	=	12.5%
Material temperature	=	630 °C
Ambient temperature	=	30 °C
Wet bulb temperature	=	28 °C
Moisture content	=	0.023 kg/kg of air
Fuel calorific value	=	10200 kcal/kg
Fuel input temperature	=	75 °C
Specific heat of fuel	=	0 5 kcal/kg °C
Temperature of combustion air	=	30 °C

ANALYSIS

Excess air in flue gas %	=	24 87
Combustion air requirement	=	13.78 kg
Total air supplied	=	17 21 kg
Total flue gas quantity	=	18.21 kg
Excess air quantity	=	3.43 kg
Heat loss due to excess air	=	354.13 kcal
H_2O vapour in flue gas		
Due to H_2 in fuel	=	1.04 kg
Due to H_2O in fuel	=	0.01 kg
Due to H_2O in air	=	0.40 kg
Dry flue gas quantity	=	16 77 kg

Appendix - 15.4/4 contd..

Kettle No.6

Basis Per kg of fuel
Procedure BIS 8753

DATA

Fuel consumption rate	=	123.5 kg/hr
Flue gas temperature	=	456 °C
CO_2 in flue gas	=	4%
Material temperature	=	365 °C
Ambient temperature	=	30 °C
Wet bulb temperature	=	28 °C
Moisture content	=	0.023 kg/kg of air
Fuel calorific value	=	10200 kcal/kg
Fuel input temperature	=	75 °C
Specific heat of fuel	=	0.5 kcal/kg °C
Temperature of combustion air	=	30 °C

ANALYSIS

Excess air in flue gas %	=	24.87
Combustion air requirement	=	13.78 kg
Total air supplied	=	17.21 kg
Total flue gas quantity	=	18.21 kg
Excess air quantity	=	3.43 kg
Heat loss due to excess air	=	306.63 kcal
H_2O vapour in flue gas		
Due to H_2 in fuel	=	1.04 kg
Due to H_2O in fuel	=	0.01 kg
Due to H_2O in air	=	0.40 kg
Dry flue gas quantity	=	16.77 kg

Appendix - 15.4/4 contd..

Surface Heat Losses

Total surface heat losses = 115625 kcal/hr

Surface losses per kg of FO supplied = 936.23 kcal

Heat loss due to door opening
(Refer Appendix - 15.4/6 for details)

Total black body radiation = 15095 kcal/hr

Radiation loss per kg of = 122.23 kcal

HEAT BALANCE

Particulars	kcal/kg	Percentage
Heat input		
Through heat value of oil	10200.00	99.78
Sensible heat in fuel	23.00	0.22
Total heat input	10223.00	100.00
Heat output		
Flue gas losses	2048.44	20.04
Heat loss due to H_2 in fuel	833.69	8.16
Heat loss due to moisture in fuel	8.05	0.08
Heat loss due moisture in air	93.49	0.91
Surface heat loss	936.23	9.16
Heat loss through door opening	122.23	1.20
Useful heat + unaccounted losses	6180.37	60.46
Total	10223.00	100.00

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Kettle No.8

Basis Per kg of fuel
Procedure BIS 8753

DATA

Fuel consumption rate	=	128.5 kg/hr
Flue gas temperature	=	520°C
CO_2 in flue gas	=	12.5%
Material temperature	=	500 °C
Ambient temperature	=	30 °C
Wet bulb temperature	=	28 °C
Moisture content	=	0.023 kg/kg of air
Fuel calorific value	=	10200 kcal/kg
Fuel input temperature	=	75 °C
Specific heat of fuel	=	0.5 kcal/kg °C
Temperature of combustion air	=	30 °C

ANALYSIS

Excess air in flue gas %	=	24.87
Combustion air requirement	=	13.78 kg
Total air supplied	=	17.21 kg
Total flue gas quantity	=	18.21 kg
Excess air quantity	=	3.430 kg
Heat loss due to excess air	=	352.7 kcal
H_2O vapour in flue gas		
Due to H_2 in fuel	=	1.04 kg
Due to H_2O in fuel	=	0.01 kg
Due to H_2O in air	=	0.40 kg
Dry flue gas quantity	=	16.77 kg

Appendix - 15.4/4 contd..

Surface Losses

Total surface heat losses = 89578 kcal/hr

Surface losses per kg of FO supplied = 697.11 kcal

Heat loss due to door opening
(Refer Appendix - 15.4/6 for details)

Total black body radiation = 15282 kcal/hr

Radiation loss per kg of = 118.93 kcal

HEAT BALANCE

Particulars	kcal/kg	Percentage
Heat input		
Through heat value of oil	10200.00	99.78
Sensible heat in fuel	23.00	0.22
Total heat input	10223.00	100.00
Heat output		
Flue gas losses	2040.11	19.96
Heat loss due to H_2 in fuel	832.75	8.15
Heat loss due to moisture in fuel	8.05	0.08
Heat loss due moisture in air	93.11	0.91
Surface heat loss	697.11	6.82
Heat loss through door opening	118.93	1.16
Useful heat + unaccounted losses	6432.45	62.92
Total	10223.00	100.00

APPENDIX - 15.4/5

WASTE HEAT RECOVERY FROM EXHAUST GASES

i. Heat in Exhaust Gases

Kettle No	Fuel consn. kg/h	Air required * kg/hr	Flue gas quantity ** kg/hr	Flue gas temp(FGT) °C	Heat in flue gas *** kcal/h
1	123 5	2125	2142	402	191237
2	127 3	2190	2207	391	191214
4	123 5	2125	2142	590	287884
6	123 5	2125	2142	456	218998
7	123 5	2125	2142	522	252927
8	128 5	2211	2228	520	262012

* Air required per kg of fuel = 17 21 kg (Refer Appendix - 15 4/4) (after considering excess air of 25%)

** Flue gas quantity = Fuel + air

*** Heat in flue gas = 0 24 x Flue gas x (FGT - 30)

ii. Heat Recovery from Exhaust gases

Heat in exhaust gases can be recovered by preheating combustion air upto 250°C by installing recuperator.

Heat Recovery

Kettle No	Heat recoverable kcal/h	Savings in FO		Cost savings Rs.lakhs/year
		kg/h	kL/year*	
1	98175	9 62	50.63	2.705
2	101178	9.92	52.21	2.790
4	98175	9 62	50.63	2.705
6	98175	9 62	50.63	2.705
7	98175	9 62	50.63	2.705
8	102148	10.00	52.63	2.812
5**	98175	9.62	50.63	2.705
Total	69401	68.02	358.00	19.13

* After considering 5000 operating hours/year

** During the audit study, kettle No.5 was not in operation hence lowest possible savings are considered.

Appendix - 15.4/5 contd..

iii. **Savings**

Total savings = 358.00 kL of FO/year

Cost savings = Rs.19.13 lakhs/year

To implement this measure the existing blowers have to be replaced with high capacity blowers of same capacity for each kettle.

Existing fuel oil consumption = 124 l/h

Air required per kg of fuel = 17.21 kg
(After considering excess air of 25%)

Density of air = 1.2 kg/m^3

Theoretical power (at 1200 mm wg) $= \dfrac{Q \times TP\ g}{3600 \times 1000}$

Q = Flow in m^3/h
TP = Total static pressure in mmwg
g = Acceleration due to gravity m/s^2

$= \dfrac{2125 \times 1200 \times 9.81}{3600 \times 1000 \times 1.2}$

= 5.8 kW

Theoretical power (at 900 mmwg) $= \dfrac{2125 \times 900 \times 9.81}{1.2 \times 3600 \times 1000}$

= 4.4 kW

Differential pressure of 3 mm wc is more than sufficient.

Hence, differential power consumption = 5.8 - 4.4

= 1.4 kW

Annual energy consumption = 1.4 x 5000 x 3.8

= Rs.26,600/-

Appendix - 15.4/5 contd..

	Annual cost saving/kettle i.e. for kettle No.1	= Rs.2.705 lakhs
	Net energy savings	= Rs.2.705 - 0.266
		= Rs.2.439 lakhs
	Cost of recuperator/kettle	= Rs.6.0 lakhs
	Cost of blower motor/kettle	= Rs.0.5
	Total cost	= Rs.6.5 lakhs
	Simple payback period	$= \frac{6.5}{2.439}$
		= 2.7 years
iv.	**Investment**	
	No.of recuperators reqd.	= 7
	Cost of recuperator/kettle	= Rs.6.00 lakhs
	Cost of blower motor/kettle	= Rs.0.5 lakhs
	Total investment reqd.	= Rs.44.5 lakhs
	Net energy savings	= 19.13 - 1.862
		= Rs.17.268 lakhs
v.	Simple payback period	$= \frac{44.5}{17.268}$
		= 2.6 years

APPENDIX - 15.4/6

HEAT LOSS DUE TO DOOR OPENING IN KETTLE FURNACES

i. No. of door openings in refining furnace = 1

ii. Width of refractory = 0.4 m

iii. Height of opening = 0.6 m

iv. Length of opening = 0.8 m

v. Emissivity of refractory(E) = 0.7

vi. Ratio = $\frac{0.6}{0.4}$ = 1.05

vii. Area of opening (A) = 0.6 x 0.8

= 0.48 m^2

viii. Radiation factor for ratio (RF 1.5) = 0.72

ix. Heat loss due to opening = (E xRF xBBR* xA) kcal/hr

* BBR - Black Body Radiation

Kettle No.	Temp °C	Black Body Radiation kcal/hr per m^2	Heat loss kcal/hr
1	750	58500	14152
2	870	78000	18869
4	900	78000	18869
6	820	62400	15095
7	822	62400	15095
8	840	63180	15282
Total			97362

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Appendix - 15.4/6 contd..

x. Savings by closing the doors

Kettle No.	Savings in FO		Cost savings Rs.lakhs/year
	kg/h	kL/year*	
1	1.38	7.26	0.388
2	1.85	9.74	0.520
4	1.85	9.74	0.520
6	1.48	7.79	0.416
7	1.48	7.79	0.416
8	1.50	7.89	0.421
Total		50.21	2.68

* Annual furnace oil savings are estimated by considering 5000 operating hours/year

xi. Investment required = Marginal

xii. Simple payback period = Immediate

APPENDIX - 15.5/1

INPUT MATERIALS TO ROTARY FURNACE

The input charge material to the rotary furnaces and batch times are :

Input materials

Refinery dross	= 3 T
Coke breeze	= 300 kg

Composition of inlet dross

Lead	= 70%
Copper	= 15%
Antimony	= 5%
Silver	= 0.2%
Slag	= 9.8
Output lead	= 2 MT

The composite of outlet slag

Lead	= 20%
Copper	= 50%
SiO_2, FeO, slag	= 30%

Batch lines

Heating upto 900-1000°C	= 2 hours
Lead time	= 30 min
Slag heating (upto 1100-1200°C)	= 4 hours

APPENDIX - 15.5/2

MONTH-WISE ENERGY CONSUMPTION AND PRODUCTION IN ROTARY FURNACE FOR THE YEAR 1994-95

Month	FO kL	LDO kL	Equi. FO kL	Total FO kL	Production MT	Sp.energy consn. L/MT	Operating hrs
May 94	-	10.000	9.474	9.474	17.000	557.276	202
Jun	10.000	-	-	10.000	9.000	1111..111	140
Jul	30.000	-	-	30.000	96.000	312.500	434
Aug	20.000	-	-	20 000	115.000	173.913	580
Sep	-	15.000	14.211	14.211	60.000	236.842	340
Oct	20.000	-	-	20.000	58.000	344.828	302
Nov	30.000	-	-	30.000	102.000	294.118	536:30
Dec	30.000	-	-	30.000	100.000	300.000	588
Jan 95	50.000	-	-	50.000	133.000	375.940	582
Feb	30.000	-	-	30.000	74.000	405.405	336
Mar	36.080	-	-	36.080	74.000	487.568	327
Total	256.080	25.000	23.684	279.764	838.000	333.848	-

Min FO consumption = 173.913 L/MT of lead

Max FO consumption = 1111.111 L/MT of lead

Avg FO consumption = 333.848 L/MT of lead

Conversion of LDO to FO = $\frac{\text{LDO (kL)} \times 0.85 \times 10800}{0.95 \times 10200}$

APPENDIX - 15.5/3

ENERGY BALANCE OF ROTARY FURNACE

Heat Inputs to the furnace

a. Furnace oil through burner
b. Coke breeze

Heat Outputs

a. Heat given to lead } Useful heat
b. Heat given to slag }
c. Heat given to complex reactions }
d. Surface heat losses
e. Flue gas losses

Heat Inputs

a. Heat in furnace oil

Oil flow rate	= 143 L/h
	= 143 x 0.95
	= 135.85 kg/h
Heat in furnace oil	= 135.85 x 10200
	= 1385670 kcal/h

b. Heat in coke breeze

Total coke breeze	= 300 kg/6 h batch
	= 50 kg/h
Heat in coke breeze	= 50 x 5500
	= 275000 kcal/h
Total heat input	= 1385670 + 275000
	= 1660670 kcal/h

Appendix - 15.5/3 contd..

Heat Outputs

a. Surface heat losses

	Particulars	Area m^2	Temp °C	Rad Loss kcal/h	Con Loss kcal/h	Tot loss kcal/h	kcal/h per m^2
A.	**Rotating drum**						
	Front portion	7 52	206	12159 71	10613 22	22772 93	3028.31
	Middle portion	7 52	173	8563 69	8187.03	16750.72	2227.49
	Back portion	7 52	140	5683 21	5897 90	11581 11	1540 04
	Total (A)	22.56		26406 61	24698.14	51104 75	2265 28
B.	Front side	7 00	350	36407 17	16050 93	52458 10	7494 01
C.	Back side	7 00	250	16995 67	10048.26	27043 94	3863 42
	Grand Total (A+B+C)	36 56		79809.45	50797 34	130606 79	3572.40

Total surface heat losses = 130606 kcal/h

b. Heat given to flue gases

Combustion air flow rate

i. Velocity of air = 12 m/s

ii. Suction dia of blower = 0.32 m

iii Combustion air flowrate = 0.9650 m^3/sec

= 3474 m^3/h

= 4169 kg/h

Flue gas quantity = Fuel + air

= FO+ coke + air

= 135.85 + 50 + 4169

= 4355 kg/h

Flue gas temperature = 750°C

Heat in flue gases = 4355 x 0.25 (750-30)

= 783900 kcal/h

Appendix - 15.5/3 contd..

c. Useful heat is heat given to lead, heat given to slag, heat given to reactions

Useful heat = heat output - (surface losses + flue gas losses)

HEAT BALANCE SHEET

Particulars	kcal/h	Percentage
Heat Inputs		
Heat given through fuel	1385670	83.44
Heat given through coke	275000	16.56
Total	1660670	100.00
Heat outputs		
Useful heat	746164	44.94
Surface heat losses	130606	7.86
Flue gas losses	783900	47.20
Total	1660670	100.00

APPENDIX - 16/1

MONTH-WISE SPECIFIC ENERGY CONSUMPTION IN ZINC OXIDE PLANT FOR THE YEAR 1994-95

Month	Production MT	LDO kL	Coke MT	Specific LDO consm L/MT	Specific Coke consm. kg/MT
Apr 94	2705.00	75.00	1650.00	27.726	609.982
May	1189.00	36.00	797.00	30.278	670.311
Jun	2870.00	85.00	1758.00	29.617	612.544
Jul	1627.00	44.00	779.00	27.044	478.795
Aug	2350.00	63.00	952.00	26.809	405.106
Sep	3150.00	88.00	1418.00	27.937	450.159
Oct	1735.00	47.00	824.00	27.089	474.928
Nov	1600.00	50.00	860.00	31.250	537.500
Dec	2750.00	77.00	1238.00	28.000	450.182
Jan 95	2790.00	78.00	1100.00	27.957	394.265
Feb	240.00	10.00	90.00	41.667	375.000
Total	23006.00	653	11466.00	28.384	498.392

Max.specific energy consm. = 41.667 L/MT of moore cake processed

Min.specific energy consm. = 26.809 L/MT of moore cake processed

Avg.specific energy consm. = 28.384 L/MT of moore cake processed

OBSERVED PARAMETERS OF WAELZ KILN

Parameters	12.00 hours	13.48 hours
CO_2 in flue gas	14.5	15.00
Flue gas temp.		
a. after dust chamber	360	330
b. inlet to tubular cooler	220	220
c. Bagfilter inlet temp.	150	160
Air flow rate velocity (ϕ 0.25m dia) m/s	14.02, 13.98, 14.25, 13.78	14.02, 13.98, 14.15, 13.92
Oil level (at 11.55 hrs 167.5)	167.5	147.0
Compressed air pressure kg/cm^2g	5.4	5.4
% Damper opening	50	55
Speed rpm	900	900
Oil pressure kg/cm^2g	4.5	4.5

Flue gas quantity = 40000 to 45000 m^3/h

CO_2 in flue gas = 0.5%

APPENDIX - 16/3

HEAT BALANCE OF WAELZ KILN

Heat Inputs to the Waelz Kiln

1. Heat Inputs

a. Heat value in LDO

LDO supply rate	= 160 L/h
	= 160 x 0.85
	= 136 kg/h
Heat value in LDO	= 136 x 10800
	= 1468800 kcal/h

b. Heat value in coke

Coke supply rate	= 2 MT/h
Heat value in coke	= 2000 x 5500
	= 11000000 kcal/h
Total heat input	= 12468800 kcal/h

2. Heat Outputs

a. Surface heat losses

Particulars	Area m^2	Temp °C	Rad Loss kcal/h	Con Loss kCal/hr	Tot loss kcal/h	kcal/h per m^2
A. Kiln section at every two meters						
I section	18.85	373	106629.08	61258.65	167887.73	8906.31
II section	18.85	372	105936.87	61035.49	166972.35	8857.95
III section	18.85	336	83080.15	53113.09	136193.24	7225.11
IV section	18 85	364	100513.96	59256.07	159770.03	8475.86
V section	18.85	350	91503.35	56167.78	147671.13	7834.01
VI section	18.85	340	85428.37	53982.36	139410.74	7395.80
VII section	18.85	323	75761.76	50307.68	126069.44	6688.03
VIII section	18 85	310	68906.90	47533.23	116440.13	6177.19
IX section	18.85	310	68906.90	47533.23	116440.13	6177.19
I section	18.85	272	51341.24	39611.29	90952.53	4825.07

Appendix - 16/3 contd..

Particulars	Area m^2	Temp °C	Rad Loss kcal/h	Con Loss kCal/hr	Tot loss kcal/h	kcal/h per m^2
I section	18.85	264	48080.95	37981.28	86062.23	4565.64
I section	18.85	250	42715.79	35162.37	77878.16	4131.17
I section	18.85	230	35763.98	31213.13	66977.11	3553.16
I section	18.85	222	33205.57	29660.35	62865.92	3335.06
I section	18.85	208	29017.37	26382.04	55399.41	2970.79
I section	18.85	171	19581.49	20163.86	39745.35	2108.51
I section	18.85	164	18041.52	18920.42	36961.94	1960.85
I section	18.85	160	17194.12	18217.09	35411.21	1878.58
I section	18.85	152	15568.38	16826.72	32395.10	1718.57
I section	18.85	138	12936.12	14448.72	27384.84	1452.78
Total	377.00		1110113.86	779374.84	1889488.71	5011.91
B. Dust chamber						
Right Hand side	35.00	60	4620.36	4163.27	8783.62	250.96
Left hand side	35.00	60	4620.36	4163.27	8783.62	250.96
Back side	15.66	56	1757.12	1557.66	3314.78	211.67
Front side	15.66	80	3794.36	3527.52	7321.88	467.55
Top side	100.00	65	15779.18	18742.33	34521.51	345.22
Total (B)	201.32		30571.37	32154.04	62725.41	311.57
Grand Total (A+B)	578.32		1140685.24	811528.88	1952214.11	3375.66

Total heat losses = 1952214 kcal/h

b. Heat in flue gas

Coke consumption = 2000 kg/h

LDO consumption = 136 kg/h

Elemental composition of combustibles in fuel

Element	Coke %	LDO %	Coke kg/h	LDO kg/h	Total kg/h	%
C	55	85.9	1100	116.82	1216.824	56.97
H_2	1	13.6	20	18.50	38.496	1.80
S	0	0.5	0	0.68	0.68	0.03
O_2	8		160	-	160	7.49
Ash	25		500	-	500	23.41
N_2	5		100	-	100	4.68
H_2O	6		120	-	120	5.62
Total	100	100	2000	136.00	2136	100.00

Actual CO_2 in flue gases = 15%

Composition in Fuel

Carbon	= 56.97%
Hydrogen	= 1.80%
Sulphur	= 0.03%
Oxygen	= 7.49%
Moisture	= 5.62%
Ash	= 23.41%
Nitrogen	= 4.68%
Total	= 100.00%

Analysis

Stoichiometric Requirements - per kg of fuel

Oxygen requirement = 1.591
Combustion air requirement = 6.856 kg

Flue gas quantity

Weight bases

CO_2	= 2.091 kg
H_2O from H_2	= 0.162 kg
H_2O from moisture in fuel	= 0.056 kg
SO_2	= 0.001 kg
N_2 in combustion air and fuel	= 7.622 kg
Dry flue gas quantity	= 7.404 kg
Max. CO_2 in flue gas %	= 28.239 Wt/Wt

Volume Basis (At STP)

CO_2	= 1.060 m^3
H_2O from H_2	= 0.201 m^3
H_2O from moisture in fuel	= 0.070 m^3
SO_2	= 0.000 m^3
N_2 in combustion air and fuel	= 4.197 m^3
Dry flue gas quantity	= 5.257 m^3

APPENDIX - 16/4

USE OF FURNACE OIL IN ZINC OXIDE PLANT

Particulars	Waelz kiln	Clinker kiln
Data		
LDO consumption	109 L/h 93 kg/h	47 L/h 40 kg/h
Hourly cost of LDO Rs./h	109 x 7.31 = 797	47 x 7.31 = 344.00
Calorific value of LDO kcal/kg	10800	10800
Specific energy cost kcal/Re	10800 x 0.85 / 7.31 = 1255.8	10800 x 0.85 / 7.31 = 1255.8
Heat requirement	93 x 10800 = 1004400	47 x 10800 = 507600
Analysis		
FO calorific value kcal/kg	10200	10200
FO requirement kg/h	1004400 / 10200 = 98	507600 / 10200 = 50
FO requirement L/h	103	52.6
Hourly cost of FO Rs/h	103 x 5.344 = 550.00	52.6 x 5.344 = 281.00
Power required for FO heating-kW (0.07 kW/L)	7.21	3.7
Total cost of power Rs/h	7.21 x 3.8 = 27.4	3.7 x 3.8 = 14.0

Particulars	Waelz kiln	Clinker kiln
Total hourly cost of heating Rs/h	Fuel + Power 550 + 27.4 = 577.4	Fuel + Power 281 + 14 = 295
Total specific cost of energy kcal/Re	10200 x 98 ------------ 577.4 = 1731	10200 x 50 ------------ 295 = 1729
Hourly cost saving Rs/h	797 - 577.4 = 219.6	344 - 295 = 49
Annual cost savings @ 6960 hrs /year Rs.lakhs/year	15.28	3 41
Eq.LDO savings kL/year	1528000 ------- 7310 = 209	341000 ------ 7310 = 47
Investment reqd. Rs. lakhs	7.80	4.00
Payback period Years	0.5	1.17

Summary

Total Equivalent LDO savings = 256 kL/year

Total cost savings = Rs.18.69 lakh/year

Investment required = Rs.11.8 lakhs

Simple payback period = 0.63 year

APPENDIX - 16/5

REPLACEMENT OF PNEUMATIC CONVEYING WITH MECHANICAL CONVEYING

1. Data

Height of Silo	= 20 m
Compressed air requirement	= 70 m^3/t
Material output	= 2 t/h
Compressed air requirement	= 140 m^3/h
Compressed air pressure	= 5.2 kg/cm^2
Specific power consumption	= 9.5 kW/ (100 m^3/h)
Power requirement	= $\frac{140}{100} \times 9.5$
	= 13 kW

Analysis

2. Type of Conveyor required

For horizontal transfer	= Screw Conveyor
For vertical transfer	= Bucket Conveyor
Power consumption for screw conveying	= 2.0 kW
Power required for bucket elevator	= 5.5 kW
Total power requirement	= 2 + 5.5
	= 7.5 kW

Appendix - 16/5 contd..

3. Savings

Savings in power	= 13 - 7.5
	= 5.5 kW
Annual operating hours	= 5000 hours
Annual power savings	= 5000 x 5.5
	= 27500 kWh/year
Annual cost savings	= Rs.27500 x 2.59
	= Rs.71225/year

4. Investment required

Cost of screw conveyor	= Rs.2.50 lakh
Cost of bucket conveyor (Rs.30000/m run)	= Rs.7.50 lakh
Total investment required	= Rs.10.0 lakh

5. Simple payback period = 14.04 years

APPENDIX - 17/1

50 TPD SULPHURIC ACID PLANT - REFRIGERATION UNIT SPECIFICATIONS

Refrigeration capacity	=	4.0×10^5 kcal/h
Refrigerating medium	=	Freon 22
Flow rate	=	30 m^3/h
Water temperature Inlet/Outlet	=	12 / 17 (°C)
a. COMPRESSORS		
Total No.	=	2
No. of cylinders	=	6
BHP	=	157.5
Drive HP	=	180
RPM	=	715
b. CONDENSER		
Tube side medium	=	Water
Shell side medium	=	Freon 22
Water flow rate	=	120 m^3/h
Design water inlet temp.	=	32 °C
Design water outlet temp.	=	36.4 °C
Total heat transfer area	=	113.7 m^2

Appendix - 17/1 contd..

c. CHILLER

Type	=	Shell and Tube
Tube side medium	=	Freon 22
Shell side medium	=	Water
Water flow rate	=	80 m^3/h
Design water inlet temp.	=	12 °C
Design water outlet temp.	=	7 °C
Total heat transfer area	=	74.6 m^2

d. CHILLER WATER PUMP

Capacity	=	30 m^3/h
Differential Head	=	30 MLC
Motor HP	=	20
Material of construction	=	Cast Iron

APPENDIX - 17/2

OBSERVATION OF 50 TPD CHILLER PLANT

Date : 6.8.95 & 7.8.95

Gas Plant	6.8.95	7.8.95
Gas cooler inlet gas temperature (°C)	40	44
Gas cooler inlet gas pressure (mmwg)	25	30
Gas cooler water inlet temperature (°C)	19	21
Gas cooler water outlet temperature (°C)	13	16

Chiller Unit		
Compressor in operation	B	B
Compressor motor load (Amps)	157	180
Suction pressure (psig)	64	60
Discharge pressure (psig)	244	227

Condenser & Chiller Unit		
Chiller exit gas pressure ((psig)	244	240
Chilled water inlet pressure (kg/cm^2g)	2.8	2.5
Chilled water outlet pressure (kg/cm^2g)	1.6	-
Condenser water inlet (kg/cm^2g)	3.2	3.1
Condenser water outlet (kg/cm^2g)	1.6	1.4

SUMMARY OF OBSERVATIONS

Data	6.8.95	7.8.95
Water temperature drop across the gas cooler (°C)	6.0	5.0
Chiller exit gas pressure (psig)	244	240
Chilled water pressure drop across the chiller (kg/cm^2g)	1.2	-
Water pressure drop across the condenser (kg/cm^2g)	1.6	1.7

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APPENDIX - 18/1

SPECIFICATIONS OF DIESEL POWER HOUSE ENGINE

I. ALLEN - NEI - LTD., DIESEL SET

Name of Manufacturer : ALLEN-NEI APE LTD.
Bedford, England

Model : VS 37G-HBC

No. of Installations : 3

a. Engine Details

No. of cylinders	16
Bore dia (mm)	325
Stroke length (mm)	370
Speed (rpm)	750
Output BHP	6930

b. Salient AC Pole Generator

Type	Brushless generator with compounding
kVA	6250
P F	0.8
RPM	750
Hz	50
Volts	11,000
Amps	328
pH	3
Frame	BA SM 100-108/8

Appendix 18/1 contd..

Details	No Load	Full Load	110 % Load
Load kVA	0	6250	6875
P F	0	0.8	0.8
V	11000	11000	11550
Frequency	50	50	50
Excitation field current *	1.7	8.5	9.2
Excitation field volts *	16.32	81.6	88.32
Exciter current	4.71	2.15	2.04

3 phase Sustained short circuit current = 879 Amps (2.68 x Full load current).

II. RUSSKY DIESEL SET

Name of Manufacturer = Russky Diesel Engines, Russia

Model = 64 T

No. of installations = 2

a. Engine Details

Cylinder bore (mm)	230
Piston stroke (mm)	300
Piston speed (m/s)	8
Mean effective pressure @ rated power (kg/cm^2g)	7.17
Specific fuel oil consumption (g/bhp/hr)	174 ± 5%

Appendix - 18/1 contd..

SALIENT POLE AC GENERATOR

Type	Three phase synchronous generator of CT-II
kW	3500
Hz	50 or 60
RPM	1000 @ 50 Hz 900 @ 60 Hz
Volt	6300 or 10500 or 11000 (r.h)

APPENDIX - 18/2

ENERGY CONSUMPTION & POWER GENERATION DETAILS

Month & Year	Power Generated (kWh)						HSD consn. (kL)	Specific power generation (kWh/kL)
	DG-I	DG-II	DG-III	DG-IV	DG-V	Total		
April 94	-	-	445905	-	326700	772605	250.00	3090.42
May 94	406500	-	1002375	-	1051650	2460525	768.664	3201.04
June 94	291750	-	2847825	-	1032750	4172325	1186.960	3515.13
July 94	500	-	2163375	-	411750	2575625	718.245	3586.00
Aug 94	500	-	33750	-	12420	46670	17.280	2700.81
Sept. 94	-	-	152550	-	-	152550	40.215	3793.36
Oct. 94	500	10250	48330	-	-	59080	19.000	3109.47
Nov 94	2250	5750	338850	-	-	346850	99.000	3503.53
Dec 94	-	-	27000	-	74520	101520	31.000	3274.84
Jan 95	2250	343000	560250	-	743175	1648675	467.895	3523.60
Feb 95	-	774500	2358450	39825	1801575	4974350	1450.000	3430.59
Mar 95	-	513420	2642035	158625	2121345	5435425	1485.600	3658.74
Total	704250	1646920	12620695	198450	7575885	22746200	6533.851	-

Appendix - 18/2 contd..

DETAILS OF SELF-GENERATION COST AND APSEB ELECTRICITY

Details	1992-93	1993-94	1994-95			
A. PHYSICAL DATA						
Power generated	486.74	263.64	227.46			
Power purchased	1082.23	1233.16	1193.70			
HSD consumed	144.21	819.91	653.47			
Lube oil consumed	159.97	68.13	73.23			
B. FINANCIAL COST						
	1992-93		1993-94		1994-95	
	Rs.lakhs	Cost/kWh	Rs.lakhs	Cost/kWh	Rs.lakhs	Cost/kWh
a. Variable Cost						
HSD	829.71	1.70	557.62	2.12	505.01	2.22
Lube oil cost	61.03	0.12	29.65	0.11	31.73	0.14
Subtotal	890.74	1.83	587.27	2.23	536.74	2.36
b. Fixed Cost						
Wages & Salaries	40.77	0.08	42.25	0.16	54.18	0.24
Interest on salaries	21.44	0.04	18.81	0.07	16.26	0.07
R&M - Wages	13.06	0.03	12.12	0.05	14.34	0.06
Stores/spares	62.24	0.13	117.32	0.45	279.52	1.23
Others	9.74	0.02	10.77	0.04	10.03	0.04
Depreciation	203.63	0.42	128.91	0.49	110.90	0.49
Subtotal	350.85	0.72	330.18	1.25	485.23	2.13
c. Total Cost of	1241.59	2.55	917.45	3.48	1021.97	4.49
Purchased power cost	2001.07	1.85	2540.71	2.06	2679.83	2.24
Total cost of power	3242.66	0.70	3458.16	2.31	3701.80	2.60

APPENDIX - 18/3

DETAILS OF DIESEL GENERATOR RUNNING HOURS

Month & Year	DG - I	DG - II	DG - III	DG - IV	DG - V	Total
April 94	-	-	129.55	-	116.50	246.05
May 94	263.20	-	255.30	-	394.45	912.95
June 94	200 30	-	682.35	-	315.10	1197.75
July 94	-	-	510.15	-	130 0	640 15
Aug 94	00.20	-	8.55	-	4 45	13.20
Sept 94	-	-	36.50	-	-	36.50
Oct 94	0 15	5 45	12 05	-	-	17 65
Nov 94	1.30	3 00	73.00	-	-	77.30
Dec 94	-	-	6 45	-	21.00	27.45
Jan 95	1 45	131 20	139 45	-	193 20	465 30
Feb 95	-	490.20	606 45	13 00	573 40	1683 05
Mar 95	-	244 20	693 05	53.25	626 40	1616.90
Total	**466.60**	**874.05**	**3152.85**	**66.25**	**2374.5**	**6934.25**
Average hr per month	38.88	72 83	262.73	5 52	197.87	-
Percentage of total running hours	6 73	12 60	45 46	0 95%	34.24	-

APPENDIX 18/4

ELECTRICAL LOADING PARAMETERS OF 5 MW DG SETS
DATE 12/8/95

Measurements taken on 33 kV panel from panel meters and kW/cosϕ digital meters.

DG Set No.	Load in MW	Pf	Amp	Volt
DG - 3	3.48	0.91	71	34.2
DG - 4	3.33	0.90	67	34.0
DG -5	3.16	0.95	59	34.1

The above measurements were taken from panel meters and kW/cosϕ digital meters.

Appendix - 18/4 contd..

OBSERVATION ON FUEL CONSUMPTION & POWER GENERATION

DATE : 11.08.95

Time : 10.00 A.M. to 4.00 P.M.

Particulars	Diesel Generator No.4		Diesel Generator No.5	
	Inlet Flow Meter Reading	Outlet Flow Meter Reading	Inlet Flow Meter Reading	Outlet Flow Meter Reading
10.00 A.M.	341143	574423	480148	724754
4.00 P.M.	349752	576376	493714	781834
Difference	8609	1953	13566	7080
Fuel consumption (L)	6656		6486	
Power generation (kWh)	20400		20400	
Specific power generation (kWh/L)	3.06		3.145	

APPENDIX - 18/5

OBSERVATIONS ON PERFORMANCE OF DIESEL ENGINES

DATE : 9.8.95

Parameter	Diesel Engine No.4	Diesel Engine No.5
RPM	740	732
Total running hours	12638	16097
Start Air pressure (kg/cm^2g)	21.0	21.0
Fuel pressure (kg/cm^2g)	4.4	1.6
JACKET WATER SYSTEM		
Jacket water pressure (kg/cm^2g)	2.4	1.85
Temp. from Engine outlet (°C)	83.0	84 0
Temp before Intercooler (°C)	87.0	-
Temp Aftercooler (°C)	81.0	57 2
SOFT WATER - RAW WATER SYSTEM		
Pressure (kg/cm^2g)	2 2	2 5
Temp before lube oil cooler (°C)	-	35.4
Temp after lube oil cooler (°C)	37.0	36.0
Temp before J/W oil cooler (°C)	37.0	36.0
Temp after J/W oil cooler (°C)	48.0	48.0
EXHAUST SYSTEM TEMPERATURE (°C)		
A1	418	492
A2	396	-
A3	356	-
A4	416	314
A5	410	486
A6	466	-
A7	360	439
A8	-	498
B1	244	440
B2	206	413
B3	358	437
B4	369	-
B5	438	385
B6	448	413
B7	464	481
B8	326	-
Exhaust temp after Turbo - `A' Bank (°C)	-	340
Exhaust temp after Turbo - `B' Bank (°C)	-	360
Air inlet before intercooler - `A' Bank (°C)	149	120
Air inlet before intercooler - `B' Bank (°C)	148	110
Air inlet after intercooler - `A' Bank (°C)	46.8	-
Air inlet after intercooler - `B' Bank (°C)	48.8	46.0
Charge air pressure (kg/cm^2g)	1.46	0.8
Governor output	6.12	6.3
Load (MW)	3.74	3.66

APPENDIX - 18/6

OPERATING ONE 5 MW DG SET CONTINUOUSLY - SYNCHRONISED WITH EB SUPPLY

Data

- Self generation statement (Refer Appendix - 18/2)
- Cost of self generation and APSEB electricity (Refer Appendix - 18/2)
- Proposed additional demand requirement of 3000 kVA from APSEB. (However this has already been obtained at the time of finalisation of report)
- Record of load shedding and power failure from APSEB (Refer Appendix - 3/6)
- Plant base load requirement = 9 MW

Proposal

* One DG set of 5 MW may be continuously run at 3.5 to 4.0 MW load for plant loads and utilise the waste heat for operating the vapour absorption chilling system for electrolyte cooler.
* This may be proposed whenever plant is planning to enhance MD by another 3 MVA or above

ANALYSIS AND RECOMMENDATIONS

One DG set is normally operated for either power cut or load shedding continuously.

Average generation potential for 365 days in a year @ 3.2 MW load = 280.32 lakh kWh

Cost of generation Rs.4.00/- (considering cost of fuel stores/spares, and lube oil only)

By proposing to run one DG set uninterruptedly (whenever 60% of plant loads are online), it is possible to reduce maximum demand requirement by 3000 kVA.

Savings in demand charges per month @ Rs.140/- kVA = Rs.4.20 lakh
= Rs.50.4 lakh/annum

Appendix - 18/6 contd..

Differential cost of power generation with APSEB energy charges are taken into account. As such, it is observed from records that one DG set is required to be run (during the last three years for 6-8 months) either for power-cut situation or load restrictions imposed.

Considering that DG is operated for 8760 hours, additional diffential cost of generation will be

= Rs.(4.0 - 3.50) x (8760 - 6900) x 3200

= Rs.29.76 lakhs

APPENDIX - 18/7

OTENTIAL WASTE HEAT RECOVERY FROM DIESEL GENERATORS

ASIC DATA ..

Sl. No.	Item/Parameter	Diesel Generator No.4	Diesel Generator No.5
1.	Fuel consumption (Litres)	6656	6486
2.	Power Generation kWh	20400	20400
3.	Specific Gravity of HSD	0.85	0.85
4.	Calorific value kcal/kg	10800	10800
5.	Exit flue gas temperature °C	450	450
6.	Actual % CO_2 in flue gases (Estimated)	7	7
7.	Duration hours	6	6

ERIVED DATA

Item/Parameter	Diesel Generator No.4	Diesel Generator No.5
Theoretical % CO_2	15.5	15.5
Percentage excess air	$\frac{15.5 - 7}{7} \times 100 = 121$	$\frac{15.5 - 7}{7} \times 100 = 121$
Theoretical air kg/kg HSD	14	14
Actual quantity of air supplied kg/kg HSD	14 (1 + 1.21) = 30.94	14 (1 + 1.21) = 30.94
Mass of flue gases per kg HSD	31.94	31.94
Heat recovery potential	Mass of flue gases x Specific heat x Drop in temperature (°C) $31.94 \times 0.24 \times 6656 \times 0.85 \times \frac{(450 - 200)}{6}$ = 1807037 kcal/hr	Mass of flue gases x Specific heat x Drop in temperature (°C) $31.94 \times 0.24 \times 6486 \times 0.85 \times \frac{(450 - 200)}{6}$ = 1760884 kcal/hr
Quantity of steam which can be generated at 1 kg/cm²g Enthalpy at 1 kg/cm²g ; 646.2 kcal/kg η WHB = 90%	$\frac{1807037 \times 0.9}{646.2}$ = 2517 kg/hr	$\frac{1760884 \times 0.9}{646.2}$ 2452 kg/hr
Quantity of steam requirement per TR (kgs)	8	8
Potential tons of refrigeration which can be generated	$\frac{2517}{8}$ = 315 TR	$\frac{2458}{8}$ = 307 TR

Appendix - 18/7 contd..

CALCULATION OF ENERGY SAVINGS AND TECHNO-ECONOMICS OF PROPOSAL

i.	Annual running hours (Atleast one D.G.Set runs for about 8 months)	= 6934.25 = 6900 hrs
ii.	Quantity of electrolyte which can be cooled (taking about 280 TR refrigeration load) and a temperature drop of 6 °C)	= $\frac{270 \times 3000 \times 0.9}{6}$ = 126000 kg/hr = 126 MT/hr
iii.	Envisaged energy savings (Based on CT fan 5 energy consumption as it is having average value)	= 7.95 kW
	Annual energy savings	= 7.95 x 6900 x 3.80 = Rs.2.08 lakhs
iv.	Additional costs by due to operation of DG in excess of 6900 hours	= Rs.29.76 lakhs
	Annual savings in demand charges	= Rs.50.4 lakhs
v.	Total annual energy savings	= Rs.(2.08-29.76+ 50.4) = Rs.22.72 lakhs
vi.	Estimated budgetary investment towards one boiler, one vapour absorption refrigeration of 300 TR	= Rs.135 lakhs
vii.	Simple payback period	= $\frac{135}{22.72}$ = 5.9 Years

APPENDIX - 19/1

DISTRIBUTION OF LIGHT FITTINGS

Sl. No.	Location	GLS		FTL	ML	HPMV				HPSV			
		200 W	500 W	40 W	160 W	80 W	125 W	250 W	400 W	70 W	150 W	250 W	400 W
	BLAST FURNACE												
1.	Periphery Ground floor	-	-	-	-	-	-	7	-	1	-	-	-
2.	I Floor	-	-	-	-	-	-	4	-	-	-	-	-
3.	II Floor	-	-	-	-	-	-	2	-	-	-	-	-
4.	III Floor	-	-	-	-	-	-	2	-	-	-	-	-
5.	Sinter Storage area	-	-	-	-	-	-	2	-	1	-	-	-
6.	Roots Blower	-	-	-	-	4	-	-	-	1	-	-	-
7.	Charging area Coke & Sinter	-	-	2T	-	1	-	1	-	-	-	-	-
8.	Storage of material	1	-	-	-	-	-	-	-	-	-	-	-
9.	Slag removal Area	-	-	-	-	-	-	1	-	-	-	-	-
10.	Rotary Furnace Area	-	1	-	-	-	-	2	-	-	1	-	-
	LEAD REFINERY												
11.	Ground floor kettle blower MCC	16	-	-	-	-	-	-	-	8	-	-	-
12.	Ground floor Casting	-	-	-	-	-	-	-	-	-	-	9	-
13.	Agitator area I Floor	-	-	-	-	-	-	2	-	-	-	9	-
14.	Lead electrical office	-	-	3T	-	-	-	-	-	-	-	-	-
	GAS CLEANING												
15.	Cooling tower area	3	-	-	-	-	-	-	2	-	-	-	-
16.	Gas cleaning blower area D.C. Motor	-	-	-	-	2	-	-	3	-	-	-	-
	CHARGE PREPARATION												
17.	Sinter preparation	4	-	-	-	-	-	-	-	1	-	-	-
	DL PLANT												
18.	MCC Room	-	-	9	-	-	-	-	-	-	-	-	-
19.	DL M/c Area	2	-	-	-	-	-	6	-	-	-	-	-
20.	Charge preparation slag + stock	-	-	-	-	-	-	4	-	1	-	-	-
21.	I Floor	-	-	-	-	-	-	14	-	-	-	-	-
22.	Hammer Mill	1	-	-	-	-	-	-	-	-	-	-	-
23.	Lead Mechanical	-	-	-	-	-	-	1	-	-	-	-	-

Appendix - 19/1 contd..

Sl. No.	Location	GLS		FTL	ML	HPMV				HPSV			
		200 W	500 W	40 W	160 W	80 W	125 W	250 W	400 W	70 W	150 W	250 W	400 W
	CELL HOUSE												
24.	Outside electrolysis area	1	-	-	-	-	-	4	-	-	-	-	-
25.	Rectifier Control room	-	-	17	-	-	-	-	-	-	-	-	-
26.	Electrolyte cooler	-	-	-	-	-	-	3	-	-	-	-	-
27.	Cell house towards rectifier side	-	-	-	-	-	-	-	2	-	-	6	-
28.	Cell house towards road side	1	-	-	-	-	-	-	6	4	-	-	-
29.	Cathode charging furnace	-	1	-	-	-	-	-	2	7	-	-	-
30.	Ingot casting area	-	-	-	-	-	-	-	14	-	-	-	-
	M R S												
31.	DG Power House	-	-	4	-	-	-	-	20	-	-	-	-
32.	Control room power	-	-	80	-	-	-	-	-	-	-	-	-
33.	Transformer Yard	-	-	-	-	-	-	-	1	-	-	8	-
	WATER TREATMENT PLANT												
34.	Outside lighting	-	-	-	-	-	-	3	-	-	-	-	-
	LEACHING												
35.	Leaching mechanical road	-	-	-	-	1	-	3	-	1	-	-	-
36.	Ball mill area - I Floor	-	-	-	-	-	1	-	3	-	-	-	-
37.	Agitator area - II Floor	-	-	-	-	-	-	-	-	2	-	-	-
38.	Pachuca Area - IV Floor	-	1	-	-	-	-	-	5	-	-	-	-
39.	Main Bridge	-	-	-	-	-	-	3	-	-	-	-	-
40.	Pachuca tank	-	-	1	-	1	-	-	-	-	-	-	-
41.	Sand settler	-	-	1	-	-	-	-	3	-	-	-	-
42.	Dorr thickner	-	-	-	-	-	5	-	2	-	-	-	-
43.	Purification	-	-	-	-	-	-	-	11	-	-	-	-
44.	Purification discharge pump	-	-	-	-	-	2	1	-	-	-	-	-
45.	Pachuca discharge pump	-	-	-	-	-	-	2	-	2	-	-	-

Appendix - 19/1 contd..

Sl. No.	Location	GLS		FTL	ML	HPMV				HPSV			
		200 W	500 W	40 W	160 W	80 W	125 W	250 W	400 W	70 W	150 W	250 W	400 W
	COMPRESSOR HOUSE												
46.	Entrance	-	-	-	-	-	-	-	2	-	-	-	-
47.	Compressor House	-	1	-	-	-	-	-	11	-	-	-	-
	ACID PLANT												
48.	Acid plant area 200 TPD	-	2	-	-	-	1	-	-	1	-	-	-
49.	50 TPD Cooler area	-	-	1	-	-	-	1	1	1	-	-	-
	TAIL GAS TREATMENT PLANT												
50.	Tail Gas plant area	-	-	-	-	1	2	1	-	-	-	-	-
51.	Mercury removal plant	-	-	-	16	-	-	-	-	-	-	-	-
	COOLING TOWER												
52.	Cooling tower periphery	-	1	-	-	1	-	1	-	-	-	-	-
	ROASTER PLANT												
53.	Ground Floor	-	-	3	-	-	2	-	-	1	-	-	-
54.	First Floor	-	-	-	-	-	2	-	-	-	-	-	-
55.	Second Floor	-	-	-	-	-	3	-	-	-	-	-	-
56.	Second floor conveyor	-	-	1	-	-	1	-	-	1	-	-	-
57.	Yard	-	-	-	-	-	-	3	-	-	-	-	-
58.	D M water plant	-	-	1	-	-	-	-	2	-	-	-	-
	CENTRAL WORKSHOP												
59.	Motor rewinding	-	-	4	-	-	-	-	13	-	-	-	-

APPENDIX - 19/2

LUX LEVEL MEASUREMENTS

Sl. No.	Location	Night
	BLAST FURNACE	
1.	Periphery	50
2.	I Floor	20
3.	II Floor	40
4.	III Floor	40
5.	Sinter Storage area	80
6.	Roots Blower	40
7.	Charging area Coke & Sinter	40
8.	Storage of material	100
9.	Slag removal Area	40
10.	Rotary Furnace Area	80
	LEAD REFINERY	
11.	Ground floor kettle blower MCC	40,100
12.	Ground floor Casting	100
13.	Agitator area I Floor	100
14.	Lead electrical office	140
	GAS CLEANING	
15.	Cooling tower area	10
16.	Gas cleaning blower area D.C. Motor	80
	CHARGE PREPARATION	
17.	Sinter preparation	40
	DL PLANT	
18.	MCC Room	180
19.	DL M/c Area	40
20.	Charge preparation slag + stock	10
21.	I Floor	40 - 60
22.	Hammer Mill	80
23.	Lead Mechanical	20

Appendix - 19/2 contd.

Sl. No.	Location	Night
	CELL HOUSE	
24.	Outside electrolysis area	20 - 40
25.	Rectifier Control room	280
26.	Electrolyte cooler	80
27.	Cell house towards rectifier side	60 - 200
28.	Cell house towards road side	60 - 200
29.	Cathode charging furnace	40 - 80
30.	Ingot casting area	40 - 80
	M R S	
31.	DG Power House	100
32.	Control room power	80 - 200
33.	Transformer Yard	80 - 100
	WATER TREATMENT PLANT	
34.	Outside lighting	20
	LEACHING	
35.	Leaching mechanical road	20
36.	Ball mill area - I Floor	60
37.	Agitator area - II Floor	40
38.	Pachuca Area - IV Floor	40
39.	Main Bridge	20
40.	Pachuca tank	20 - 40
41.	Sand settler	40
42.	Dorr thickener	20 - 100
43.	Purification	100
44.	Purification discharge pump	20
45.	Pachuca discharge pump	10

Appendix - 19/2 contd..

Sl. No.	Location	Night
	COMPRESSOR HOUSE	
46.	Entrance	20
47.	Compressor House	100
	ACID PLANT	
48.	Acid plant area 200 TPD	40
49.	SO_2 Blower	40
50.	50 TPD Cooler area	20
	TAIL GAS TREATMENT PLANT	
51.	Tail Gas plant area	10 - 40
52.	Mercury removal plant	40
	COOLING TOWER	
53.	Cooling tower periphery	20
	ROASTER PLANT	
54.	Ground Floor	20
55.	First Floor	40
56.	Second Floor	60
57.	Second floor conveyor	60
58.	Yard	20
59.	D M water plant	60
	CENTRAL WORKSHOP	
60.	Motor rewinding	40

APPENDIX - 19/3

OBSERVATIONS OF LIGHTING - KEPT SWITCHED ON EVEN DURING DAY TIME (IN OUTDOOR YARD/AREAS)

Sl. No.	Area	Type of fitting	Date
1.	Yard in front of control lab safety dept.	2 x 400 W, MV Flood	
2.	Roaster Towers	MV 250W MV	27.7.95
3.	Leaching Yard	MV 250 W	27.7.95
4.	Cell House	10 New fittings around building	26.7.95 27.7.95
5.	Cell House (Pumps area) (64A, 77, 78 Pumps)	2 x 250 W ML Lamps on (Ordinary holder)	27.7.95 27.7.95 27.7.95
6.	ZNO Plant	4 x 125 W MV 1 x 100 W Inc.	27.7.95 27.7.95
7.	Lead refinery building Canteen	2 x 250 W SV 1 x 125 W SV	27.7.95 27.7.95
8..	Leaching area next to transformer S/S	1 x 250 W SV	27.7.95
9.	Scrubber area - Roaster	250 W MV Lamp on	27.7.95
10.	Compressor - PCC	1 x 300 W	01.8.95
11.	Electrolyte coolers	All light in outdoor yard `ON'	01.8.95

APPENDIX - 19/4

REPLACEMENT OF INCANDESCENT LAMPS BY HPSV LAMPS

A. Replace 200 W GLS by 70 W HPSV

Nominal luminous flux of 200 W GLS	=	3040 Lumen
Nominal luminous flux of 70 W HPSV	=	5800 Lumen
No. of 200 W GLS in the plant	=	29
Power consumed by 29 nos. of 200 W GLS	=	29 x 200
	=	5800 W
Power consumed by 29 ncs. of 70 W HPSV inclusive of ballast loss	=	(70 + 30) x 29
	=	2600 W
No. of operating hours	=	12 hrs/day
Energy consumed by 29 nos. of GLS per year (330 working days)	=	5.8 x 12 x 300
	=	20,880 Kwh/year
Energy consumed by 29 nos. of 70 W HPSV lamps per year	=	2.6 x 12 x 300
	=	10,440 Kwh/year
Energy saved/year	=	10,440 Kwh
Cost savings	=	Rs.26,518/-
Cost of implementation @ the rate of Rs.550/- per set	=	Rs.15,950/-
Simple payback period	=	8 months

Appendix - 19/4 contd..

B. Replace 500 W GLS by 150 W HPSV

Nominal luminous flux of 500 W GLS	=	8200 Lumen
Nominal luminous flux of 150 W HPSV	=	13500 Lumen
No. of 500 W GLS in the plant	=	7
Power consumed by 7 nos. of 500 W GLS	=	7 x 500
	=	3500 W
Power consumed by 7 nos. of 150 W HPSV inclusive of ballast loss	=	(150 + 30) x 7
	=	1260 W
No. of operating hours	=	12 hrs/day
Energy consumed by 7 nos. of GLS per year (330 working days)	=	3.5 x 12 x 300
	=	12600 kWh/year
Energy consumed by 7 nos. of 150 W HPSV lamps per year	=	1.26 x 12 x 300
	=	4536 kWh/year
Energy saved/year	=	8064 kWh
Cost savings	=	Rs.20482/-
Cost of implementation @ the rate of Rs.1750/- per set	=	Rs.12250/-
Simple payback period	=	8 months

APPENDIX - 19/5

REPLACEMENT OF HPMV 250 W BY 150 W HPSV

Nominal luminous flux of 250W HPMV lamp	= 13,500 lumens
Nominal luminous flux of 125W HPSV lamp	= 13,500 lumens
No.of 250W HPMV lamps in the plant	= 71
Energy consumed by 71 nos. of 250W HPMV lamps (12 hrs/day for 300 working days)	= 250 x 12 x 300 x 71 = 63,900 kWh/year
Energy consumed by 71 nos. of 150W HPSV lamps	= 150 x 12 x 300 x 71 = 38,340 kWh/year
Energy savings/year	= 63,900 - 38,340 = 25,560 kWh/year
Cost savings/year	= Rs.64,920
Cost of implementation @ Rs.1750/- per set	= Rs.1.25 lakhs
Simple payback period	= 2 years

Appendix 19/5 contd..

REPLACEMENT OF HPMV 400 W BY 250 W HPSV

Nominal luminous flux of 400W HPMV lamp	=	25,000 lumens
Nominal luminous flux of 250W HPSV lamp	=	23,000 lumens
No.of 400W HPMV lamps in the plant	=	65
Energy consumed by nos. of 400W HPMV lamps (12 hrs/day for 300 working days)	=	400 x 12 x 300 x 68
	=	97,920 kWh/year
Energy consumed by of 250W HPSV lamps	=	250 x 12 x 300 x 65
	=	61,200 kWh/year
Energy savings/year	=	97,920 - 61,200
	=	36,720 kWh/year
Cost savings/year	=	Rs.93,270/-
Cost of implementation @ Rs.2750/- per set	=	Rs.187,000
Simple payback period	=	2.1 years

APPENDIX - 19/6

REPLACEMENT OF HPMV BY HID METAL HALIDE LAMPS

CENTRAL WORKSHOP

Length L	= 72 mt
Breadth W	= 18 mt
Height H	= 14 mt
Mounting height HM	= 13.5 mt
Reflector factor ceiling	= 0.3
Reflector factor wall	= 0.1
Reflector factor floor	= 0.1
Maintenance factor Mf	= 0.7
Lux Eav	= 200 lux

ALTERNATIVE - I

High Intensity Discharge Metal Halide Lamps

Nominal luminous flux at 100 hours of 400 W HID metal Halide Lamps	= 50,000 lumens
Rated average burning hours	= 10,000 hours
Luminaire	= High bay

Design Calculations

Room index $\dfrac{L \times W}{Hm \times [L+W]} = \dfrac{72 \times 18}{(13.5) \times (72 + 18)} = 1.1$

Utilisation factor UF = 0.8

No. of luminaires required $= \dfrac{Eav \times L \times W}{Lumens \times Uf \times Mf}$

$= \dfrac{200 \times 72 \times 18}{50,000 \times 0.8 \times 0.7}$

$= 9$

Capital Cost

Cost of HID Metal Halide lamp (with Ignitor) = Rs.2250

Appendix - 19/6 contd..

∴ Total cost = Rs.2250 x 9

= Rs.20,250

Energy consumed by the lighting load of the shop at the rate of 16 hours/day for 300 working days.

$$= \frac{400\ W \times 16 \times 300}{1000} \times 9$$

= 17,280 kWh/year

Present Status :

400 W HPMV Lamp

No of HPSV luminaires	= 13
Wattage	= 400 W

∴ Total energy consumed at 16 hours/day for 300 working days

$$= \frac{400 \times 13 \times 16 \times 300}{1000}$$

= 24960 kWh/year.

Energy saved by use of Metal Halide lamp	= 24960 - 17280
	= 7680 kWh/year
Cost savings	= Rs.19,500/-
Cost of implementation	= Rs.20,250/-
Simple payback period	= 1.1 years

APPENDIX - 19/7

USE OF VOLTAGE CONTROLLERS IN DIFFERENT AREAS FOR LIGHTING

SILVER REFINERY

Connected load (assuming 70% utilisation) = 2.96 kW

15% of energy can be saved using voltage controllers.

Energy saved = 2.96 x 0.7 x 0.15

= 0.31 kW

Energy saved per year (@ 12 hours/day for 300 working days) = 1118.88

Cost savings = Rs.4251/-

Cost of implementation = Rs.5032/-

Simple payback period = 1.18 years

Sl No	Area	Connected Load (kW)	Energy (kWh/yr)	Cost (Rs.)	Cost of implementation (Rs.)	Simple payback period (years)
1	Silver Refinery	2 96	1118.88	4,252	5,032	
2	Gas cleaning plant Lead	5 96	2252.88	8,561	10,132	
3	Slurry neutralisation plant Lead Plant	1 17	442.26	1,681	1,989	
4	Sinter House Lead Plant	13.6	5140.8	19,535	23,120	
5	Burner House	2.24	846.72	3,218	3,808	
6	Dross Treatment Plant Zinc casting	4.72	1784.16	6,780	8,024	
7	Zinc Casting plant LP 17L	15.68	5927 04	22,523	26,656	
8	MCC Room Clinker Kiln Bin	1.7	642.6	2.442	2,890	
9	Clinker kiln outdoor feed house and dust chamber	2.64	997.92	3,792	4,488	
10	Lead Plant Blast Furnace	7.7	2910.6	11.060	13,090	
11	Lead Plant Crusher house	6.64	2509.92	9,538	11,288	
12	Lead Plant Storage & charge Preparation	11 52	4354.56	16.547	19,584	

Appendix - 19/7 contd..

Sl No	Area	Connected Load (kW)	Energy (kWh/yr)	Cost (Rs.)	Cost of implementation (Rs.)	Simple payback (years)
13	Lead Plant Storage & charge preparation	14 36	5428 08	20,627	24,412	
14	WAELZ kiln	1 84	695 52	2,643	3,128	
15	Lead Plant Lead Refinery	14.68	5549 04	21,086	24,956	
16	Clinker Kiln Bag filter	0.6	226.8	862	1,020	
17	Zinc oxide coke storage and handling	16.77	6339 06	24,088	28,509	
18	Clinker Kiln Tubular cooler	1 82	687 96	2,614	3,094	
19	Bag filter house and cadmium storage	1 6	604 8	2,298	2,720	
20	Tubular cooler feeder system Pneumatic I & II	2 84	1073 52	4,079	4,828	
21	Cadmium plant	9 83	3715 74	14,120	16,711	
22	Sludge Drying & filtration plant	6 3	2381.4	9,049	10,710	
23	Electrolyte cooling tower L 19	1 22	461 16	1,752	2,074	
24	Pump house Water Treatment Plant	2 5	945	3,591	4,250	
25	Pump House Cooling tower area	2 84	1073.52	4,079	4,828	
26	LDO Area MCC Room	0 4	151 2	575	680	
27	Fuel Oil MCC Room	0 92	347 76	1,321	1,564	
28	SS, Leaching & Compressor House	2.32	876 96	3,332	3,944	
29	S S, Roaster & C T	2.24	846.72	3,218	3,808	
30	SS, Workshop	1.16	438 48	1,666	1,972	
31	SS Lead Plant	1 32	498 96	1,896	2,244	
32	SS Zinc Oxide Plant	1.16	438.48	1,666	1,972	
33	Cell House FF	9.68	3659.04	13,904	16,456	
34	Cell House GF	12.24	4626.72	17,582	20,808	
35	Leaching plant FF	13.43	5076.54	19,291	22,831	
36	Leaching plant GF	9.8	3704.4	14,077	16,660	
37	Raw water sumps Pump house	0 865	326 97	1,242	1,471	
38	Compressor House	2 98	1126 44	4,280	5,066	

APPENDIX - 20/1

MONTH-WISE CATHODE PRODUCTION AND POWER CONSUMPTION

Month	Production (MT)	Power Consumption (kWh)
May 94	6196	12012
June 94	7493	13603
July 94	6866	14503
Sept.94	7602	13590
Oct. 94	3845	9390
Total	32002	63098

Specific power consumption $= \frac{63098}{32002}$

$= 1.97$ kWh/MT

APPENDIX - 20/2

MEASURED CELL VOLTAGES AND MILLI VOLT DROPS

DATE : 17.08.95

Cell No.	Cell Voltage Anode to Cathode
10	2.35
9	2.34
8	2.35
7	2.36
6	2.37
5	2.35
4	2.36
3	2.33
2	2.35
1	2.34
Total	23.5

Cell No.1	Anode Milli Volt Drops
1	3.65
2	5.60
3	9.35
4	9.05
5	3.20
6	3.53
7	1.80
8	6.23
9	5.30
10	2.57
11	2.05
12	2.28
13	3.84
14	1.82
15	1.89
16	3.45
17	6.36
18	2.52
29	3.40
20	4.60
21	11.5
22	4.20
23	5.11
24	5.60
Average	4.54

APPENDIX - 2

LIST OF SUPPLIERS AND RETROFITS

Eqpt./Retrofit	Manufacturer
Lighting	Beblec (India) Ltd 126, Sipcot complex Hosur 635 126, Tamilnadu
	Electronics India 238/A, 10th Main Road Nagendra Block, BSK II Stage Bangalore 560 050
Voltage Controllers	Beblec (India) Ltd 126, Sipcot complex Hosur 635 126, Tamilnadu
	Electronics India 238/A, 10th Main Road Nagendra Block, BSK II Stage Bangalore 560 050
Capacitors	Asian Electronics Ltd D-11, Road No.28 Wagle Industrial Estate Thane - 400 604
	Marketed by : Mysore Sales Intl.Ltd Industrial Products Dvn. MSIL House, 36, Cunningham Road Bangalore 560 052
	Meher Capacitors Pvt Ltd 16(K), Attibele Industrial Area Neralur 562 107 Bangalore District.
	Marketed by : Larsen & Toubro Limited P O Box 119, Pune 411 001

Eqpt./Retrofit	Manufacturer
	Prabhodan Capacitors Mfg.by Seva Engg.Works Saswadi, Pune
	Crompton Greaves Ltd Dr.E.Moses Road Worli, Bombay 400 018
Energy efficient Motors	Siemens Limited Jyothi Mahal II Floor St.Marks Road, Bangalore 560 001
	Crompton Greaves Limited Machine I Division Dr. E Moses Road Worli, Bombay 400 018
Soft Starters	Jeltron Instruments (I) (P) Ltd 6-3-248/F Road No.1 Banjara Hills Hyderabad 500 034
	Jayshree ElectroDevices (P) Ltd 101, Prabhodhan Apartment 64/9, Erandewane, Pune 411 004
	Bharat Bijilee Ltd Industrial Electronic Division 501-502, Swastik Chambers Chembur, Bombay 400 071
Variable Speed Drives	Asea Brown Boveri Ltd Sona Towers, 71, Miller Road Bangalore 560 052
	Kirloskar Electric Co Ltd Unit-IV, Belawadi Indl Area Mysore 510 005

Eqpt./Retrofit	Manufacturer
	Siemens Limited Jyothi Mahal, III Floor -49 St.Marks Road, Bangalore 560 001
	Allen Bradley Ltd C-11, Site-4 Industrial Area, Shahibad Pin 201 010
	Integrated Engineering Systems 135, Damji Shamji Indl. Estate LBS Marg Vikroli, Bombay 400 083
	MAN Industries B143 Sanjay 5, Mittal Estate Andheri East Bombay 400 009
Insulating material	Lloyd Insulation Pvt Ltd H-13, Connaught Circus New Delhi 110 003
	Hyderabad Industries Ltd Sector 25, Faridabad -121 004 Haryana
	Orient Cerawool Ltd 5, Basement, Dalamal Towers Nariman Point Bombay 400 021
Water Flow & Oil Flow Meters	Kent Meters Ltd Agent : L & T Limited Gulas Bhavan 6, Bahadur Shah Zafar Marg New Delhi 110 002
	Eureka Industrial Eqpmt Pvt Ltd 258, Kalina Udyog Bhavan Prabhadevi Bombay 400 025

Eqpt./Retrofit	Manufacturer
	Rojaram Consultants A, 5 Surabhi Apartments 21, Abhiramapuram, I Street Madras 600 018
	SS Engineering Industries H-5, South Extension Part I, New Delhi 110 049
	Rockwin India E 1/7, Vasant Vihar New Delhi 110 057
Automatic damper control for CO_2	JNM Systems & Services P B No.37 Bombay-Pune Road Kasarwadi Pune 411 055
	Taylor Instruments Co.(I) Ltd 14, Mathura Road P O Amar Nagar Faridabad
	Industrial Business Associates Shop No.6, Ratan Sadan Sane Guruji Road Jacob Circle, Bombay 400 063
Power Analyser (To measure kVA, kW, PF, V & A)	Microtek Instruments 40-A, I Main Road I Floor, CIT Nagar Madras 600 035
Recuperators	Wellmann Incandescent (I) Ltd 7, Pretoria Street Calcutta 700 001
	Encon Furnaces (P) Ltd 14/6, Mathura Road Faridabad - 121 003

Eqpt./Retrofit	Manufacturer
	Heat Recovery Division Thermax Limited Chinchwadi, Pune 411 019
	Incorporated Engineers Ltd Shri Ram Chambers RC Dutt Road Baroda 390 005
	L & T Limited 1B, Park Plaza, 171, Park Street, Calcutta 700 016
Lux Meter	Cocin Prakrito Instrumentation 16, Rajendra Nagar P O Mohan Nagar Ghaziabad 201 007
Anemometer	Microtech Instruments 40-A, I Main Road CIT Nagar, Madras 600 035
O_2 & CO_2 Analysers	J N Marshall Systems & Services P B No.37, Bombay Pune Road Kasarvadi, Pune 411 005
	Taylor Instrument Co (I) Ltd 14, Mathura Road PO Amarnagar, Faridabad
Star Delta Auto Controllers	Project & Supply A-605, Sunswept, Lokhandwala Comple Swami Samarth Nagar Four Bunglow, Andheri (W) Bombay 400 056
	Technovation Control & Power System 5, Savita Sangam Society Near Rajesh Apartment, Gotri Road Baroda 390 007

Eqpt./Retrofit	Manufacturer
Compact Fluorescent Lamps	GE-Apar Lighting Maker Chambers 111, I Floor Nariman Point Bombay 400 021
	Crompton Greaves Ltd. Lighting Division Dr E Moses Road Worli Bombay 400 018
Fyrite Kit For CO_2 Measurement in flue gases	J N Marshall Pvt Ltd Kasarwadi, Poona 411 034 Maharashtra
Furnace Curtains	Urja Products Pvt Ltd 423, GIDC Telephone Exchange Lane Odhav, Ahmedabad 382 415
Cooling Towers	Paharpur Cooling Towers 81/B, Diamond Harbour Road Calcutta 700 027
	Mihir Engineers Pvt. Ltd. 3rd Floor, Dr D N Road G.P.O. Box No.1389 Bombay 400 011
Air Preheater	Reliance Heat Transfer Pvt.Ltd. 46, Veer Nariman Road, Fort Bombay 400 023
	Wester Works Engineers Pvt.Ltd. Commerce Centre J Dadajee Road Bombay 400 034
Flash Steam Generation	J N Marshall Pvt. Ltd. Bombay Poona Road Kasarwadi, Poona

Eqpt./Retrofit	Manufacturer
Steam Traps	Harfort Manufacturing C-204, Akshay, Y A C Nagar Kondivita Road Bombay 400 059
	Uni Klinger Ltd Liberty Building Sir Vittal Thakersay Marg Bombay 400 020
	Hawa Engineers Pvt Ltd 29-A, Highway Commercial Centre Dani Limda, Ahmedabad 380 028
F R P Blades for Cooling Towers	Parag Enterprises Pvt Ltd 43, Tarani Colony, AB Road Dewas, Madhya Pradesh 455 001
Calcium silicate blocks	Hyderabad Industries Ltd Sector 25, Faridabad -121 004 Haryana
Vapour Absorption Refrigeration Machine	Thermax Ltd Chinchwadi, Pune 411 019
Refrigerated air dryer	PACE Equipment (P) Ltd. 27 Q & R Laxmi Indl. Estate New Link Road, Andheri (West) Bombay 400 058
	Indican Polymer Ltd. 311, Manasaravor 90, Nehru Place New Delhi 110 019
	Trident Industries Ltd. 408 Sathy Road Ganapathy, Coimbatore 641 006

t./Retrofit	Manufacturer
ricators of Copper ercell bus bars, kibles	AVI Machine Spares Off.B/24 Vraj Villa Ist Floor Amrutnagar, LBS Marg, b Ghatkopar Indl. Estate Ghatkoar (W) Bombay 400 086 **Factory 1 :** B-15/a Ghatkopar Indl. Estate, LBS Marg, Ghatkopar (W) Bombay 400 086 **Factory 2 :** 3, Hanuman Indl.Estate Kastak Road, Wadala Bombay 400 031 **Factory 3 :** 886/c, GIDC Indl.Estate Makarpura Baroda 390 010
	Mistry Prabudas Marji, Op Mithal Industrial Estate, Andheri Kura Road Bombay 400 059 Ph : 6341828 / 6343860
	B M Moonot & Neutronics Mfg. Co. Regd.Office Station View Bldg. Chembur, Bombay 400 071 Works : 12-1 Marol Maroshi Road Opp.State Bank of India, Andheri (East) Bombay 400 059 Tel. 583749